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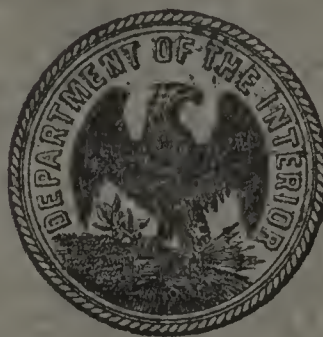
THE ELECTRIC FURNACE IN
METALLURGICAL WORK

BY

DORSEY A. LYON, ROBERT M. KEENEY

AND

JOSEPH F. CULLEN

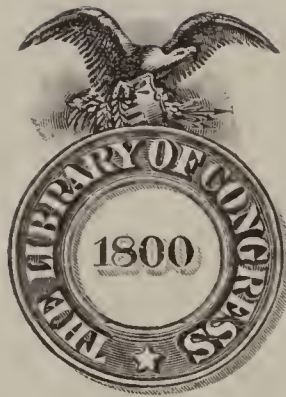


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U. S., BUREAU OF MINES

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BY

DORSEY A. LYON, ROBERT M. KEENEY

AND

JOSEPH F. CULLEN



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THE ELECTRIC FURNACE IN METALLURGICAL WORK.

By DORSEY A. LYON, ROBERT M. KEENEY, and JOSEPH F. CULLEN.

PART I. DESIGN, CONSTRUCTION, AND OPERATION OF ELECTRIC FURNACES.

By DORSEY A. LYON and JOSEPH F. CULLEN.

INTRODUCTION.

In connection with its investigation looking to the prevention of waste and the increase of safety and efficiency in the mineral industries the Bureau of Mines has undertaken a study of the possible uses of the electric furnace in metallurgy. This bulletin presents a compilation of such data as seem to be useful for the information of persons who may be interested in the matter. In the application of electricity to metallurgy three kinds of processes are involved, namely, mechanical, thermal, and electrolytic.

Under the mechanical application of electricity come those processes of dressing and concentrating ores that are known as either electrostatic or electromagnetic processes.

In electrothermic processes the current is used as a source of heat. The fundamental differences between the various types of electric furnace that are at present in commercial use relate to the manner in which the electric current is applied, as may be noted by reference to that part of this report in which the subject is discussed.

In electrolytic processes the electric current is used for depositing metals from solutions. The process may be conducted in one of the following ways: (1) The material to be treated may be put into solution and then electrolyzed, the material thus treated forming the electrolyte; (2) the material to be treated may form the anode or be placed in contact with the anode; or (3) the material to be treated may form the cathode or be placed in contact with the cathode.

This report treats of electrothermic processes, with the possible exception of those for the reduction of aluminum from its ores. Moreover, of the processes used in the so-called electrochemical industries only those for the reduction of an ore to metal, or to a compound of the metal that is of value in itself, or is an aid in metallurgical work, are here discussed.

DEVELOPMENT OF USE OF ELECTRICITY IN METALLURGICAL WORK.

The use of electricity as a source of heat in metallurgical work is comparatively recent. Although many chemists and metallurgists had previously conducted small experiments, Siemens, in 1882, was probably the first to make use of it in metallurgical work for commercial purposes.^a Then came the furnace of E. H. and A. H. Cowles,^b who in 1885 invented a furnace for the production of aluminum alloys and for a variety of other purposes. A year later Paul Héroult^c and C. M. Hall^d patented processes for the production of aluminum. In 1893 Wilson,^e aided by data gathered by Moissan in his classical experiments, developed the calcium-carbide furnace, which was followed in 1896 by Acheson's carborundum furnace.^f Two years later Captain Stassano of Italy patented an electric furnace^g for smelting iron ores, and the following year demonstrated the working of his process. In 1900 the production of ferro-alloys in the electric furnace was begun, and at the present time practically all ferro-alloys are made in electrically heated furnaces.

As a result of the work of Stassano and of the successful making of ferro-alloys in the electric furnace, steel was next made in France by Héroult and in Sweden by Kjellin under patents taken out in 1900. In 1906 experiments on the reduction of iron ores in the electric furnace were conducted by the Canadian Government and by Grönwall, Lindblad, and Stalhane in Sweden. The production of pig iron in the electric furnace in Sweden at the present time is a direct result of the work of the three men last mentioned.

In the development of the electric furnace its uses have been extended, and to-day it is used not only in metallurgy but in many branches of applied chemistry as well.

FIELD OF THE ELECTRIC FURNACE IN METALLURGY.

The electric furnace is not necessarily a competitor of the combustion furnace, for it has won its position in the industrial world chiefly through the fact that it permits the use of a higher temperature than can be attained with the combustion furnace. This is why, for example, high-grade ferrosilicon and many of the ferro-alloys are not produced in the combustion furnace. Again, the electric furnace is particularly suited to the production of those elements that

^a Siemens, William, and Huntingdon, —, Report of 52d meeting: British Assoc. Adv. Science, August, 1882, pp. 496-498.

^b U. S. patent 319795 (1884).

^c French patents 175711, April 23, 1886; and 170003, April 15, 1887.

^d U. S. patents 400766 and 400664, April 2, 1889 (applied for July 9, 1886).

^e Wilson, T. L., *Industries and Iron*, vol. 20, 1896, p. 322.

^f Fitzgerald, F. A. J., *The carborundum furnace: Electrochem. Ind.*, vol. 4, 1906, p. 53.

^g *Electrochem. and Met.*, vol. 1, 1901, p. 230; *Electrochem. Ind.*, vol. 1, 1902-3, pp. 247-363.

may themselves constitute the walls of the furnace, so that it is possible to effect a great saving in crucibles and retorts, the cost of which was a large item in older methods. As the best illustration of such products, phosphorus and carbon bisulphide may be cited.

In short, the advent of the electric furnace into metallurgy has made possible developments that were not possible through the use of the combustion furnace. For example, such substances as magnesia, lime, and molybdenum were formerly considered infusible, but at the high temperature attained in the electric furnace these substances are not only melted but may be volatilized. New processes based on this development have been perfected for the purification and separation of substances, and metals and their combinations may be separated by fractional distillation in the same general way as substances having comparatively low boiling and volatilizing points.

It is perhaps true that some metallurgists have unduly emphasized the importance of the electric furnace in metallurgical work, and for that reason it may be well briefly to consider a few of the fundamental facts regarding the possibilities of the use of high temperatures.

In the first place, it is thought that any known oxide can be reduced by carbon within the range of temperature possible in the electric furnace. At any rate, it is easy in such a furnace to reduce such stable oxides as those of calcium, silicon, and magnesium, and to produce carbides of these elements as well as those of boron, aluminum, molybdenum, tungsten, and titanium. These carbides are extremely stable, resisting not only attacks of water, but also most of the active chemical agents.

The writers have purposely emphasized the fact that the electric furnace is not a competitor of the combustion furnace, but has its own particular field of usefulness. However, quite possibly it may in time become a competitor of the combustion furnace for certain work, and, in fact, it already is to some extent.

COMPARISON OF THE ELECTRIC FURNACE AND THE COMBUSTION FURNACE.

METHOD OF PRODUCING HEAT IN THE TWO FURNACES.

As is well known, in the combustion furnace heat is produced by the oxidation of some combustible substance, the product of combustion being a gas. The efficiency attained, or the ratio between the amount of heat usefully employed in the furnace and the heat value of the fuel or electric energy supplied, is generally rather low, a fair average being probably about 25 per cent. On the other hand, the source of heat in an electric furnace is electrical energy, and it is perhaps fair to assume that the average efficiency of such a

furnace is 60 per cent, whereas in the combustion furnace it is perhaps never more than 66 per cent, as shown in the following tables:

Average thermal efficiency of various types of combustion furnaces.

Type of furnace.	Product.	Thermal efficiency. ^a
		<i>Per cent.</i>
Blast.....	Cast iron.....	52 to 66
Acid process.....	Steel.....	11.9
Basic process.....	do.....	10.0
Reverberatory.....	Pig iron.....	8.5
Do.....	Wrought iron.....	5
Siemens crucible.....	Steel.....	4
Greenwood crucible.....	do.....	2
Retort.....	Zinc.....	2 to 3

^a Average efficiency for steam, 50 to 60 per cent.

Average thermal efficiency of three types of electric furnace.

Type of furnace.	Product.	Horse-power.	Temperature.	Thermal efficiency.
			[°] C.	<i>Per cent.</i>
Acheson.....	Graphite.....	1,000	3,300	75
Jacobs.....	Fused Al ₂ O ₃	200	-----	74
Acheson.....	Carborundum.....	1,000	3,000	76.5

COMPARISON OF HEATING EFFICIENCY.

Some idea as to the heating value of electrical energy as compared to that of coal may be obtained from the following discussion of “Electric heat versus heat from fuel.” ^a

One kilowatt-hour ^b is equivalent to 860 kilogram-calories, which is approximately the full heat obtained from 100 grams of good coal when completely burned to carbon dioxide. If the cost of a ton of coal of 2,000 pounds (907 kg.) is *a* dollars, and if the efficiency of heat production by burning coal is *x* per cent, then the coal required for producing 860 kilogram-calories costs *a* divided by 90.7 *x* dollars. On the other hand, if the cost of 1 kilowatt-hour is *b* cents and if the efficiency of producing heat from electrical energy is *y* per cent, then the kilowatt-hours required for producing 860 kilogram-calories cost *b* divided by *y* dollars. Hence electrical heat will be cheaper than heat produced by combustion of fuel if *a y* is greater than 90.7 *b x*. To go further, we need the efficiency figures *x* and *y*. By assuming definite figures we introduce, of course, an uncertainty into our comparison, but it will probably be considered fair if, for a first approximation, we assume a 25 per cent efficiency for fuel heating and a 75 per cent efficiency for the electric furnace. Then we conclude that electrical heat will be cheaper than heat produced from fuel if *a* is greater than 30.2 *b*, or, in other words, if the cost of a ton of coal in dollars is more than 30 times the cost of a kilowatt-hour in cents. For instance, to compete with coal at \$6 per ton, the electric kilowatt-hour would have to cost less than 0.2 cent. This is clear evidence that if the electric furnace did not have other important features it would not compete with fuel heat under ordinary conditions.

^a Electrochem. and Met. Ind., vol. 5, 1907, p. 298.

^b For definition of electrical units, see p. 186.

We have purposely made this comparison on the basis of the ton of coal and of the kilowatt-hour, although it has become customary to use the electric horsepower-year as the unit of electrical energy in such estimates. Then the above condition can be stated in this way: Electric heat will be cheaper than fuel heat if the cost of 1 ton of coal is more than one-half of the cost of the electric horsepower-year. Stated in this form, the comparison looks more favorable for the electric furnace than when stated as in the preceding paragraph. But in reality it is not proper to make the comparison on the basis of a horsepower-year, since we thereby implicitly assume that the electric furnace is working continuously every hour all year round, which is in general a decidedly improper assumption.

It is, of course, clear that the above comparison is exceedingly narrow in that it considers only one single side of the problem, namely, the amount of fuel and electrical energy required to produce the same number of calories. But even in this very respect the above comparison falls short of the truth and does not do justice to the electric furnace. The chief reason is that electrical heating is essentially internal heating, permitting a very high concentration of energy at any point wanted and thus enabling one to produce high temperatures. Fuel heating, on the other hand, is always more or less transmission of heat from the fuel to the charge, and the rate of heat transmission depends on the difference of the two temperatures; this rate decreases rapidly the higher we go up in the temperature. Naturally with fuel heating we can never obtain any higher temperature than that of the burning fuel itself. There is no corresponding limitation in electric heating. This is the fundamental reason why for all very high temperature processes the electric furnace reigns supreme.

LIMITATIONS TO ELECTRIC HEATING.

It is necessary to qualify the statement that there is no limitation to electric heating. Theoretically there is not, but practically there is, for the following reasons: As an electric current passes through a conducting medium heat is produced. The intensity of this heat depends on the amount of current that passes, and as most substances are conductors when hot, the degree of intensity possible is theoretically unlimited. In practice, however, the conducting substance begins to fuse when heated to its melting point, and one is then confronted with the physical difficulty of keeping the conducting medium in place. Or if this be accomplished, the conducting medium ultimately vaporizes, the gaseous materials escape, and heat is thus carried away from the furnace as rapidly as it is supplied. The temperature of the electric arc, which is somewhere between $3,600^{\circ}$ and $4,000^{\circ}$ C., is perhaps the highest temperature attainable at present.

Theoretically it is possible to obtain by the combustion of carbon in oxygen a higher temperature than can be attained in the electric furnace. In practice, however, the attainment of such temperature is impossible for the following reasons:

1. At high temperatures carbon oxidizes to CO and not to CO₂. The oxidation of 1 pound of carbon to CO₂ generates 8,100 pound-calories, whereas the oxidation of 1 pound of carbon to CO generates only 2,430 pound-calories.

2. The gaseous products formed escape rapidly and carry heat away from the furnace.

3. In commercial work, at present at least, it is not feasible to supply pure oxygen for combustion purposes.

4. Atmospheric oxygen is accompanied by four times its volume of nitrogen, so that when air is used instead of pure oxygen the nitrogen is raised to a high temperature and carries from the furnace the heat it has absorbed.

SUMMARY.

The foregoing paragraphs may be briefly summarized, as in the following sentences:

In the combustion furnace, no matter what form of fuel is used, the temperature can not exceed $2,000^{\circ}\text{C}$.

In the combustion furnace the efficiency varies from 2 per cent to 60 per cent, whereas in the electric furnace it averages about 60 per cent.

In the combustion furnace not only is it necessary to take care of the products of combustion, but these products may reduce the efficiency of the process, whereas the electric furnace is free from these difficulties.

In the electric furnace the temperature attainable is the same whether the furnace is operated on a commercial or on a laboratory scale.

In the electric furnace, the heat may be developed at any desired point, and by passing the current through the substance that is to be treated the heat necessary for the operation may be developed within the substance itself.

The temperature of an electric furnace is under absolute control.

Any atmosphere desired—that is, neutral, oxidizing, or reducing—may be obtained within the electric furnace. This feature, together with the fact that the product is generally much purer than that of the combustion furnace, is a great factor in favor of the electric furnace.

In general, one may say that it is necessary to use an electric furnace in all operations requiring a temperature higher than $2,000^{\circ}\text{C}$.; for work requiring a temperature of less than $2,000^{\circ}\text{C}$., local conditions and the nature of the work to be done will affect the choice of the electric or the combustion furnace. As has been previously pointed out, the cost of power will largely enter into the matter, but even with a high cost of electric power, the superior efficiency of the electric furnace, the possibility of obtaining a purer product, and perhaps greater safety for the workmen—any or all of these factors may lead to a decision in favor of the electric furnace.

CLASSIFICATION OF ELECTRIC FURNACES.

Electric furnaces may be classified in a more or less arbitrary manner, and different authorities have used such classifications as seemed most desirable. Stansfield's classification ^a is substantially as follows:

I. Arc furnaces:

- A. Those with independent arc (Moissan furnace).
- B. Those with direct-heating arc (Siemens vertical-arc furnace).

II. Resistance furnaces:

A. Those with special resistor—

- 1. Furnaces in which charge constitutes the resistance (carborundum furnace).
- 2. Furnaces having resistor in walls (electrical tube furnace).

B. Those without special resistor—

- 1. Furnaces in which electrolysis is not employed (charge constitutes the resistance):
 - (a) Those in which the charge is not melted, but remains in a solid condition (graphite furnaces).
 - (b) Those in which charge is added in solid condition but subsequently becomes liquid (Hérault ore-smelting furnaces).
 - (c) Those in which charge is added in liquid condition (Kjellin steel furnaces).
- 2. Electrolytic furnaces (furnaces for producing aluminum).

Burgess ^b classifies electric furnaces according to the character of the medium constituting the conductor, as follows:

I. Furnaces in which the heat is developed by the passage of current through a solid conducting medium or "resistor."

A. The conducting medium, or core, may consist of the material that is to undergo useful transformation.

B. The heat is developed in a core of conducting material and this heat in turn is communicated to the surrounding material that constitutes the charge, either with or without a wall between the two cores and the charge.

II. Furnaces in which the heat is developed by the passage of current through a solid or liquid conducting medium.

A. Electrolytic.

B. Nonelectrolytic.

III. Furnaces in which the heat is developed by the passage of current through a gaseous conducting medium.

A. Arc furnaces.

(1) In which an arc plays between two or more electrodes in the neighborhood of material to be treated.

(2) In which an arc is maintained between one carbon electrode and the material to be treated, the latter acting as a second electrode.

RESISTOR FURNACES.

A resistor signifies a substance that is used as a medium for developing heat when electric current is passed through it. As stated by Burgess in his classification of electric furnaces, a resistor may be

^a Stansfield, A., *The electric furnace, its evolution, theory, and practice*, 1907, p. 32.

^b Burgess, C. F., *The present status of electric furnace working*: Jour. Western Soc. Engrs., vol. 10, 1905, p. 167.

either a solid, a liquid, or a gas. Materials commonly used for this purpose are the metals, carbon, and substances that are not conductors at ordinary temperatures, but become conductors when hot. When such substances are used, it is necessary first to heat them by some auxiliary means. The construction of a resistance furnace is based upon the fact that by resistance an electric current is transformed into heat.^a

SOLID RESISTOR FURNACES.

FURNACES DEVELOPING HEAT IN THE CHARGE ITSELF.

Furnaces in which the heat is developed by passage of the current through a solid conducting medium are described below.

An illustration of the type of furnace in which the conducting medium, or core, may consist of the material that constitutes the charge is the Acheson graphite furnace, shown in figure 1.^b In this furnace a core, *c*, of carbon rods is needed to carry the current until

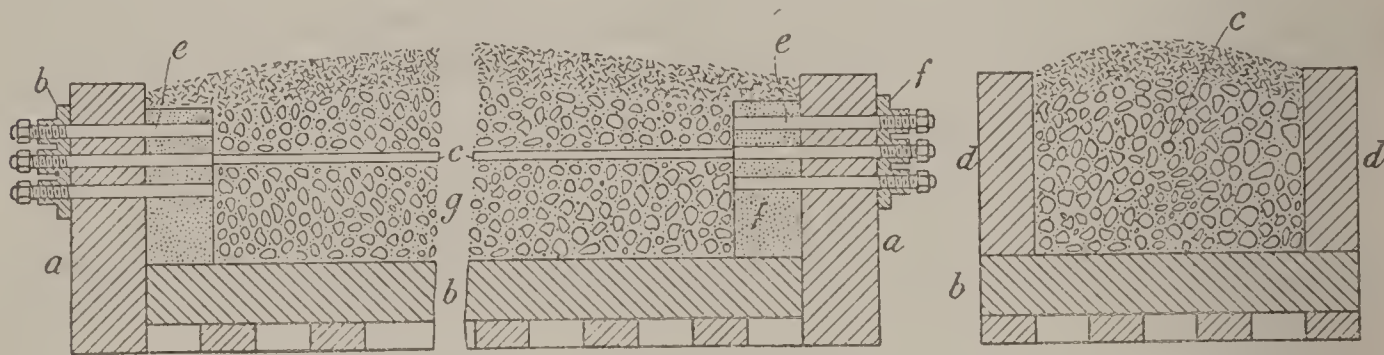


FIGURE 1.—Acheson graphite furnace.

the charge becomes heated. The base, *b*, and end walls, *aa*, which support the electrodes, are permanent, but the side walls, *dd*, are not, and can be pulled down after a run. The electrodes comprise a number of graphite rods, *e*, which are set in a block of carbon as shown in the figure; electric contact being made by a terminal plate, which may be water-cooled. In making graphite in a furnace of this kind a charge of anthracite coal, *g*, is placed around the core of carbon rods, *c*, and then covered with a layer of material that is a poor conductor of both heat and electricity. When the current is turned on, it passes through the core of carbon rods and heats the anthracite, which is gradually converted into graphite by the high temperature attained. The electrical resistance of the graphite is very low, and for that reason the current has little tendency to pass through the outer parts of the charge.

FURNACES DEVELOPING HEAT IN A CORE.

That type of furnace in which the heat is developed in a core of conducting material, this heat being in turn communicated to the surrounding material that constitutes the charge, may or may not have a wall separating the core from the charge.

^a For laws of electrical resistance, see p. 187. ^b Stansfield, A., *The electric furnace*, 1907, p. 145.

FURNACES WITH A WALL BETWEEN CORE AND CHARGE.

The electrically heated tube furnace (fig. 2) is an example of the type of furnace having a wall between the core and charge. As is shown in the figure, it is constructed by winding a wire, *r*, which forms the resistor, around a tube, *t*, which is made of porcelain or some other highly refractory substance. The resistor wire may be of either platinum or an alloy, such as nickel and chromium, that has a high fusing temperature. In some furnaces of this type, the resistor wire, instead of

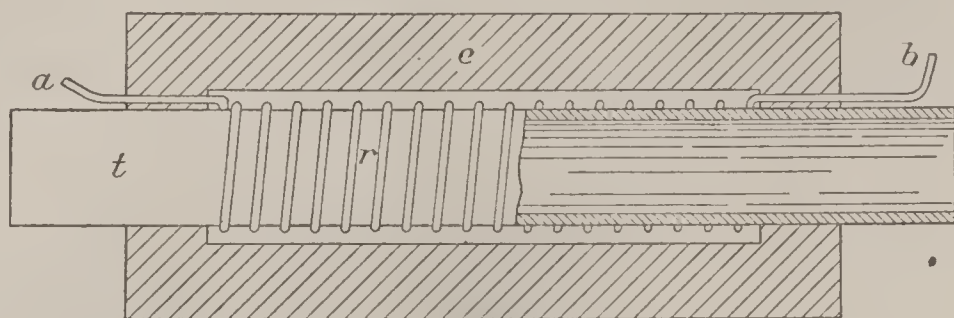


FIGURE 2.—Cross section of electrically heated tube furnace.

being wound around a tube, is embedded in the inner surface of the refractory envelope, *e*. In the operation of the tube furnace the charge is placed in the tube, the leads *a* and *b* are connected to an electric circuit and, by means of a controlling rheostat, enough current is supplied to the resistor wire to overload it, so to speak, and the resistance that the current meets transforms the electric energy into heat. By varying the intensity of the current it is possible to obtain any desired temperature below the melting point of the wire, which for platinum is 1775° C. or 3227° F.

FURNACES WITHOUT WALL BETWEEN CORE AND CHARGE.

The Acheson carborundum furnace,^a shown in figure 3, is an example of the type of furnace in which there is direct contact between the core and the charge.

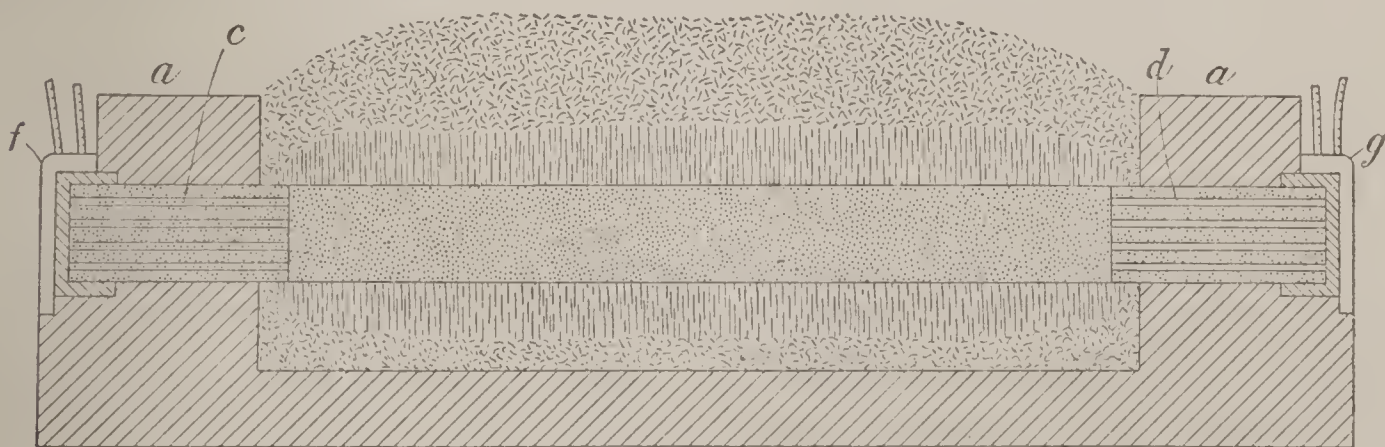


FIGURE 3.—Acheson carborundum furnace.

The end walls *aa* are permanent, and support the heavy bronze electrode holders *f* and *g*. Inserted into these holders are large bundles of carbon rods, *c* and *d*, and between the rods is a core of broken carbon which remains until the end of the operation. Around this core is placed a mixture consisting approximately of 34.2 per cent coke, 54 per cent sand, 9.9 per cent sawdust, and 1 per cent salt.

^a Fitzgerald, F. A. J., The carborundum furnace, *Electrochem. Ind.*, vol. 4, 1906, p. 53.

The broken carbon core acts as the resistor. When a current is passed through this core it becomes heated and in turn heats the surrounding charge, which then becomes a conductor and acts as a resistor itself.

FURNACES IN WHICH THE RESISTOR IS A LIQUID.

Furnaces in which heat is developed by the passage of a current through a liquid conducting medium are of two types, electrolytic and nonelectrolytic, to use terms already defined.

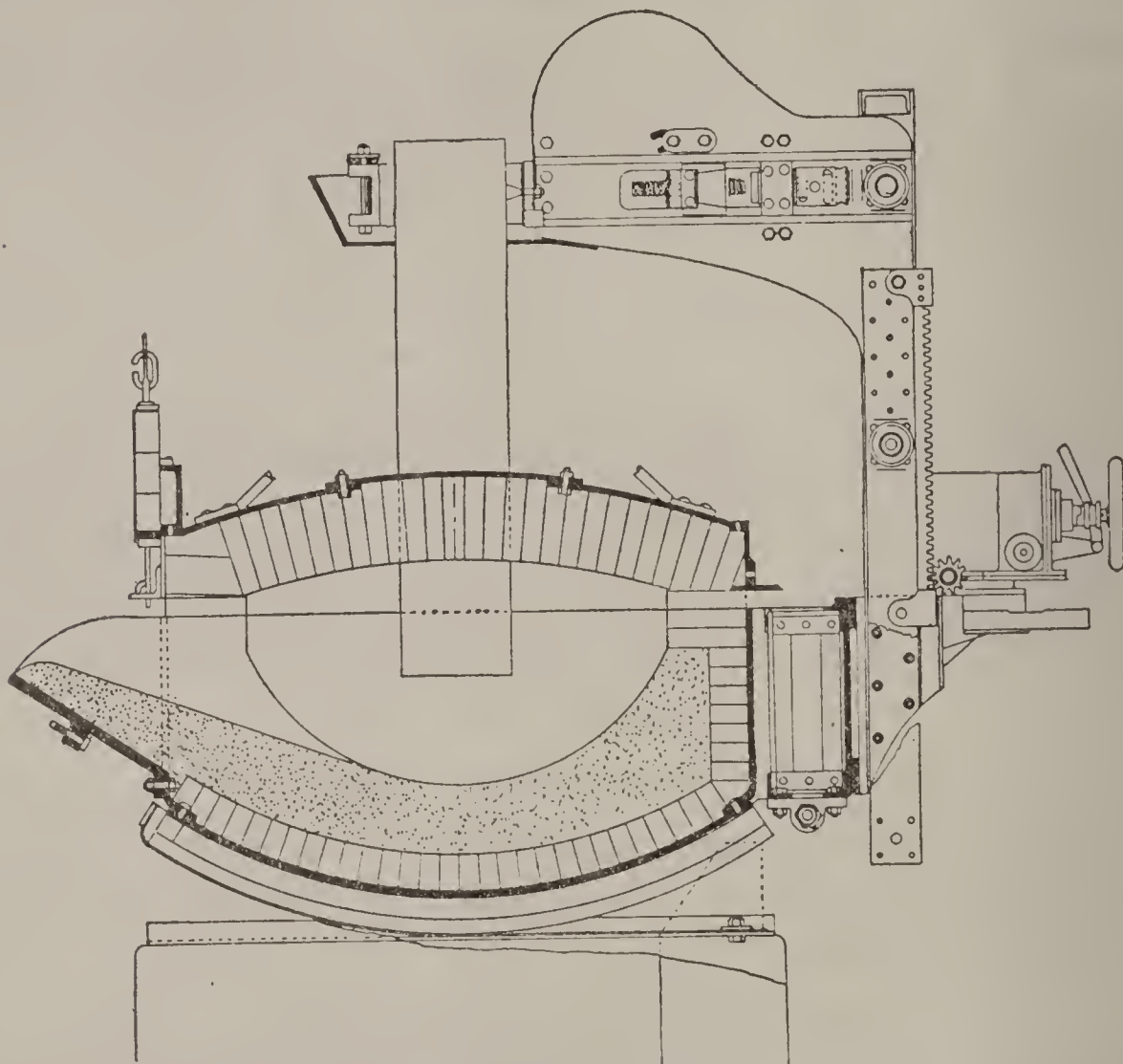


FIGURE 4.—Héroult 2.5-ton, single-phase steel furnace.

ELECTROLYTIC FURNACES.

The furnaces in which aluminum is made by the Héroult and Hall processes are the best examples of the electrolytic type of furnace. Inasmuch as their construction is described in the second part of this bulletin, a description is not necessary here.

NONELECTROLYTIC FURNACES.

The Héroult steel furnace (fig. 4) is illustrative of the nonelectrolytic type of furnace. In this furnace the molten steel is covered with a layer of slag. The heat in the furnace is generated by the passage of the electric (alternating) current between the electrodes through the liquid layers of slag and metal.

ARC FURNACES.

Arc furnaces may be classified, according to the method used in forming the arc, in two general types: (1) Furnaces in which the arc plays between two or more electrodes in the neighborhood of the

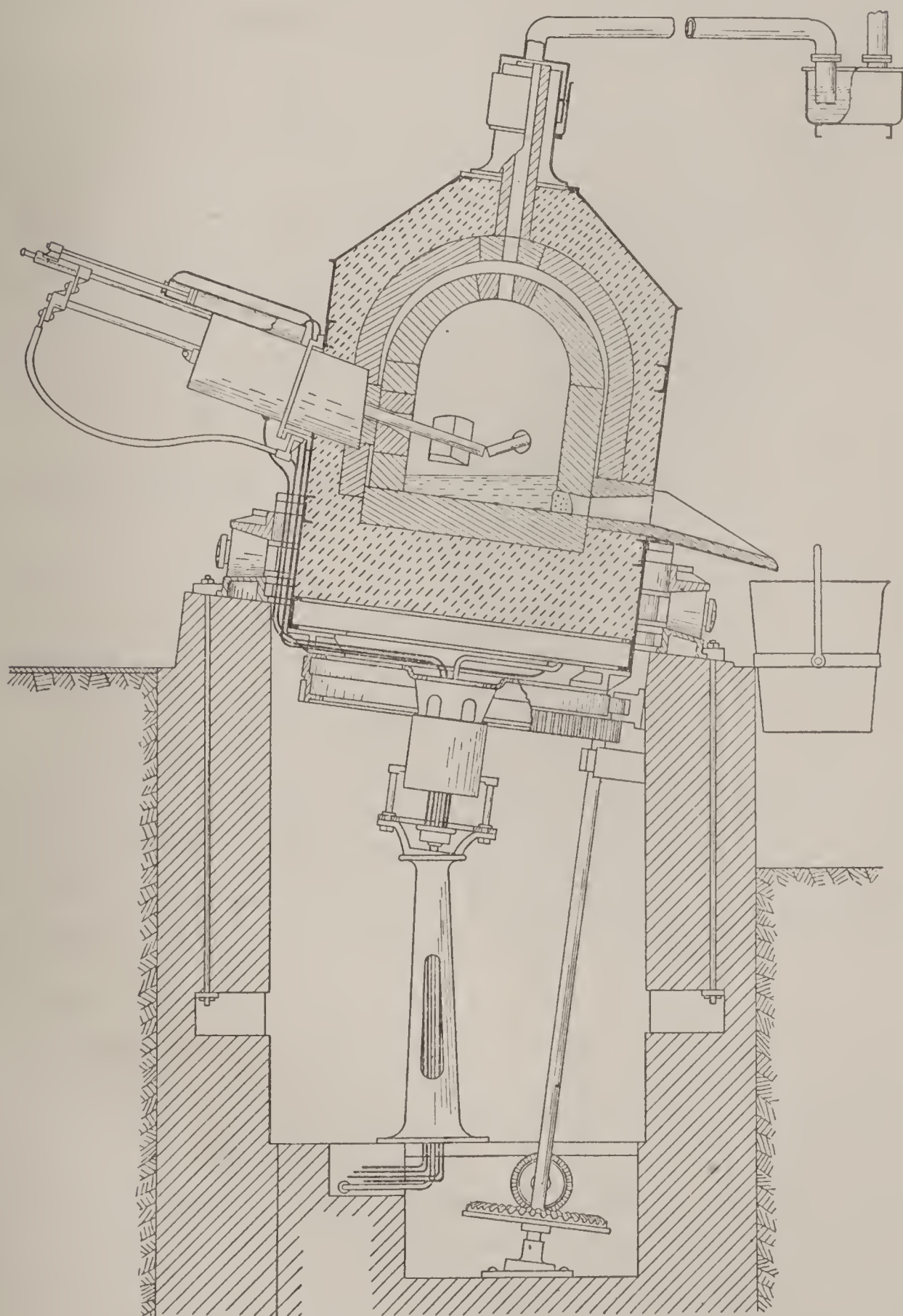


FIGURE 5.—Stassano furnace.

material to be treated, and (2) furnaces in which the arc is maintained between one carbon electrode and the charge, the latter acting as a second electrode.

The Stassano furnace (fig. 5) is a good illustration of the first type. An arc is maintained between the ends of two electrodes, and

in this way the metal and slag are heated by the heat radiated from the arc.

The Siemens vertical-arc heating furnace, as shown in figure 6, is a furnace of the second type. In this furnace the charge *d*, which is in contact with the lower electrode *c*, is heated by maintaining an arc between the end of the upper electrode *b* and the charge *d*.

FACTORS GOVERNING DESIGN AND CONSTRUCTION OF ELECTRIC FURNACES.

Naturally, in the design and construction of an electric furnace, the aim is to obtain in the operation of the furnace the highest effi-

ciency compatible with the operating conditions. Some of the factors that govern the efficiency of an electric furnace are briefly considered below.

RELATION BETWEEN ELECTRIC ENERGY AND HEAT.

The measure of the electric power supplied to the electric furnace is the watt. If 1,000 watts be used for one hour, one kilowatt-hour has been used. The expenditure of this much energy is equivalent to 860 kilogram-calories, which represents the full amount of heat to be expected from the complete combustion (to carbon dioxide) of 100 grams of good coal. By calculation it is possible to determine the amount of heat or the temperature necessary to bring about the results desired in any metallur-

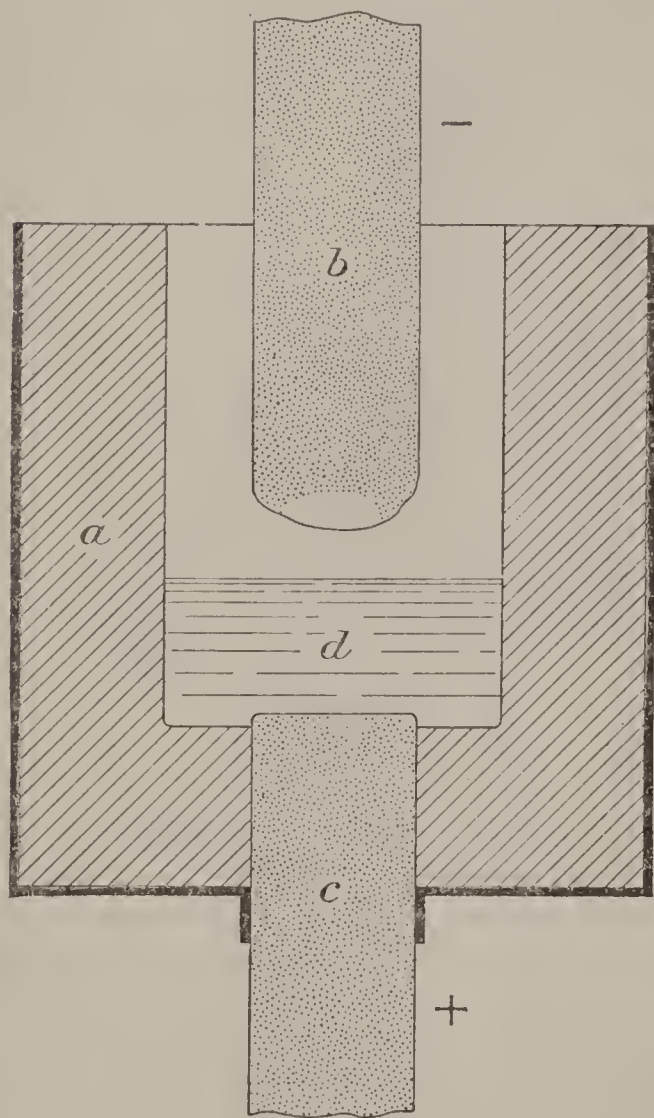


FIGURE 6.—Cross section of Siemens vertical-arc heating furnace.

gical process. To attain and maintain this temperature a certain rate of heating is necessary, this rate depending on the rate at which the heat is conducted away through the furnace walls, the dimensions of the furnace, and the cooling effect of the charge that is added from time to time. When, as in the electric furnace, this heat is to be supplied by electrical energy, the first consideration is so to construct the furnace as to obtain the largest practicable percentage of heat value from the electrical energy supplied. In order to do this, the designer must have, of course, a clear understanding

of the fundamental laws of electricity^a and the heating effects of the electric current. The different methods of heating may be briefly stated as follows: Direct heating by resistance; induction heating; indirect resistance heating; and arc heating. The methods are briefly described below.

METHODS OF HEATING WITH ELECTRICITY.

DIRECT HEATING BY RESISTANCE.

An illustration of direct heating by resistance is the heating of a metal bar or wire by passing an electric current through it. The metal will become hot because of its inherent resistance. Increasing the current increases the resistance and consequently the temperature. As can be readily understood, the disadvantage of direct heating by resistance is that not only is the metal or substance heated, but likewise all the conductors that carry the current to it. For this reason copper, which has a much lower specific resistance than iron,

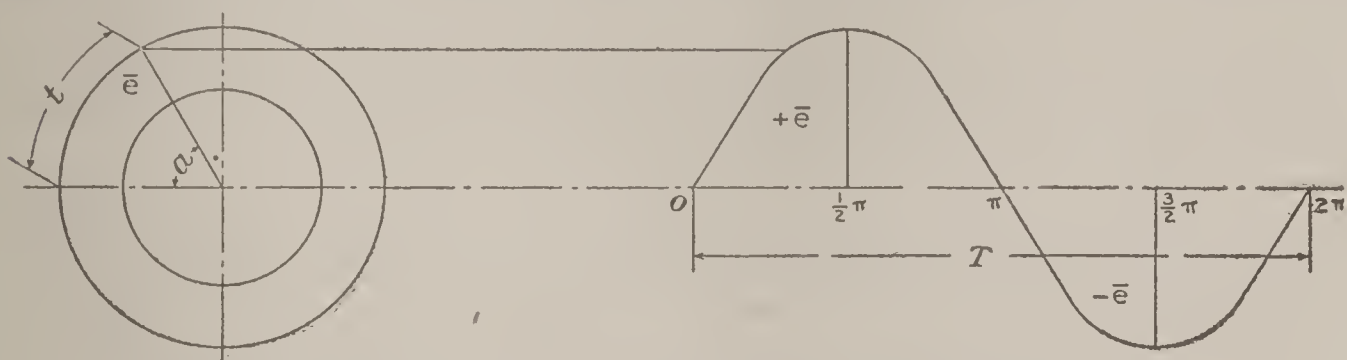


FIGURE 7.—Curve representing a complete wave or cycle of alternating current.

is used for a conductor. If an electric current be passed through a copper and an iron wire, each of equal cross section, greater heat is developed in the iron wire, in the proportion of 0.1 to 0.018; hence if the wires are of equal lengths the total amount of heat developed in the iron wire will be much greater than that developed in the copper wire.

INDUCTION HEATING.

In order to define precisely what is meant by induction heating, it is necessary first to outline the principles upon which such heating is based. As is well known, in a direct electric current the flow is always in the same direction, whereas in an alternating current the direction of flow is constantly being reversed. The time required for a change of direction is called a period. In figure 7, which shows a complete wave or cycle, T represents the time necessary for the current to swing through a complete wave; that is, from 0 (zero) to the positive maximum, back to zero, to the negative maximum, and back to zero. If an alternating current be passed through a coil of wire and if a conductor be passed through the magnetic field

^a For the convenience of those not familiar with the same, some of the fundamental laws and terms of electrical engineering are given in the Glossary (pp. 186-189).

of the coil in such a way as to cut the lines of force of the field a current is induced in the conductor. A magnetic field is shown in figure 8 in which n and s are the poles of a magnet and c is a wire conductor (shown in cross section) that is made to cut the lines of force passing between the poles.

In what is known as a transformer the conductor is kept stationary, and the field is made to change or to move across the conductor. If a coil of wire be wound around a core of soft iron and a current passed through the wire, a magnetic field is produced as with a permanent magnet. If the current be suddenly made and broken or if an alternating current be used, the magnetic flux will increase and decrease in first one direction and then the other. Now by winding a second coil of wire on the iron core a conductor is obtained. Every time the magnetic flux (lines of force of the field) increases or decreases in strength as it changes its direction a current will be set up or induced in the second coil or secondary winding and in a direction

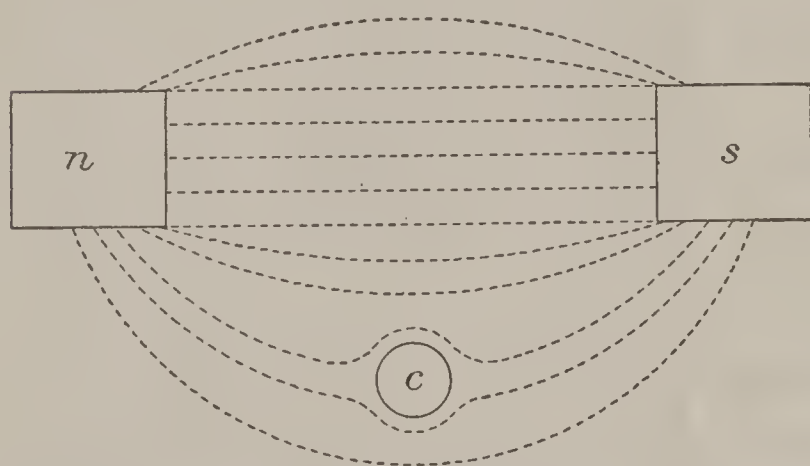


FIGURE 8.—A magnetic field, in which a conductor has been placed.

that tends to oppose the flow of current through the first coil or primary winding.

The current induced in the secondary winding will be inversely proportional to the number of turns on the winding and consequently

(by Ohm's law, $\frac{E}{I} = R$)

the voltage will be in direct proportion. The foregoing relation does not hold exactly true because the resistance and other properties of the transformer become factors in the application of this law.

With a given voltage the larger the current to be carried the lower must be the resistance of the conductor. The resistance can be lowered by using a conductor of larger size or one made of a material that is a better conductor of electricity. In electrical work copper is generally used for conductors.

From these statements it is evident that heat is generated when an electric current is induced in a conductor, and if the conductor has a high specific resistance, such as that of the bath of steel in an induction furnace, the heat developed is in proportion to the square of the resistance and to the strength of the current induced. This method of heating may be termed heating by induction, and it is most used in electric-furnace practice in steel furnaces, such as those of Kjellin, Colby, and Röchling-Rodenhauser, which are known as induction furnaces.

CONSTRUCTION OF THE INDUCTION FURNACE.

On analyzing the induction furnace its method of construction is found to be based on three essential features, as follows: (1) A primary winding composed of a great many turns of wire or other conductors, (2) a core or circuit for the magnetic flux, and (3) a circular trough containing molten steel, which corresponds electrically to a secondary winding of one turn.

As the secondary winding is of only one turn, an enormous current can be induced in it, and as the steel is not an exceptionally good conductor it becomes heated to a high temperature. For example, in such a furnace as the Kjellin steel furnace the primary winding represents the conductor, through which is passed a current of 90 amperes at 3,000 volts. Around this winding is a circular trough which contains the metal. As the current passes through the primary winding, a magnetic flux, or flow of current, is induced in the metal in the trough, and the current is transformed from 90 amperes at 3,000 volts to 30,000 amperes at 7 volts. By reason of the high amperage of the induced current and the resistance with which it meets in the metal in the trough, sufficient heat to melt the metal is generated.

Although this type of furnace possesses certain advantages, it also possesses several disadvantages, chief among these being that it is inefficient in much the same sense as a poorly constructed transformer is, owing to the fact that it is built on the principle of a transformer. Moreover, when used for heating a metal such as steel, there is also induced in the bath of the induction furnace electro-mechanical forces, which are discussed in another chapter, that have a tendency to interfere with the working of the furnace.

INDIRECT-RESISTANCE HEATING.

In the indirect-resistance method of heating the heat is not produced by using a metal as a resistor, but by passing the electric current through some substance that has a very low electric conductivity. An example of such a furnace is the Acheson carborundum furnace (fig. 3). In this furnace the charge to be heated is placed around the carbon core. An electric current is passed through the core, which thus becomes hot and in turn heats the charge.

When materials such as crucible steel are heated in crucibles, an indirect method of heating is employed. The crucible furnaces are fired with either coal or gas and have low efficiency (see table, p. 4), owing to the amount of heat that is carried away with the flue gases. The Girod crucible furnace ^a (fig. 9) was devised to do the same kind of work as the ordinary crucible steel furnace. In the

^a Vom Baur, C. II., *Electric furnaces in the iron and steel industry*, 1913, p. 33.

Girod furnace shown in the figure the current is conducted through the bottom, which is made of materials having high electrical resistance, such as carbon and silica. In furnaces of this kind temperatures

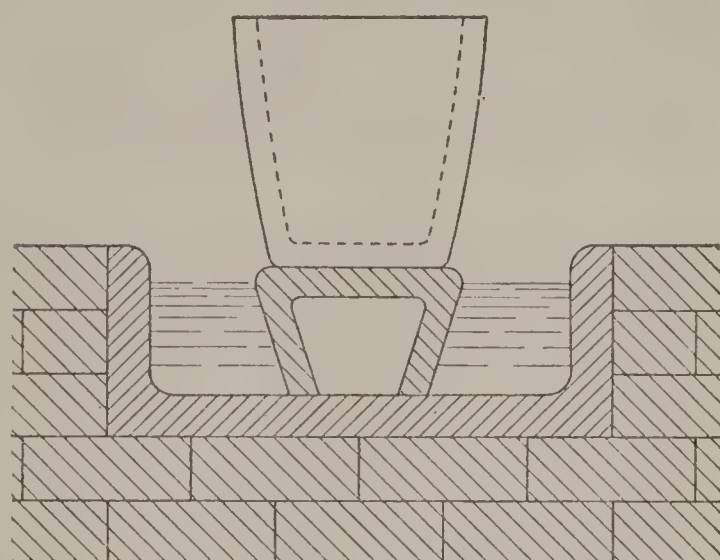


FIGURE 9.—Cross section of Girod furnace.

of $1,400^{\circ}$ to $1,700^{\circ}$ C. have been reached. In another Girod crucible resistance furnace a resistor of ferrosilicon is used, to which the current is led by four carbon blocks around the crucible.

Still another type of indirect-heating furnace is the Helberger furnace (fig. 10). This furnace consists of a crucible which is placed in circuit by means of carbon contacts, so that the current passes vertically through the crucible walls. Helberger uses the ordinary carbon or graphite crucibles.

ARC HEATING.

Arc heating was employed by Moissan in his famous researches. In figure 11 is shown the Moissan type of furnace, which is described by Stansfield^a as follows:

It consists of two blocks of limestone and two carbon rods, to which electrical connections are made. A cavity is hollowed out in these blocks, and the material to be heated is placed in a crucible of carbon or magnesia. As even lime melts and volatilizes at the temperature of this furnace, a lining of alternate layers of carbon and magnesia was arranged as shown in the figure, in order to withstand, as far as possible, the heat of the arc.

In some of these experiments Moissan converted two or three hundred electrical horsepower into heat in a furnace of only a few inches internal dimensions. At the enormously high temperature of his furnace everything melts or turns to vapor. Carbon is the most refractory substance known, and even that turns to graphite and volatilizes; magnesia, another very refractory substance, melts at the highest temperature of the furnace and vaporizes. Lime, quartz, and alumina melt and boil in the furnace. Gold, copper, iron, and in fact all the metals, can also be melted and boiled in such an electrical furnace.

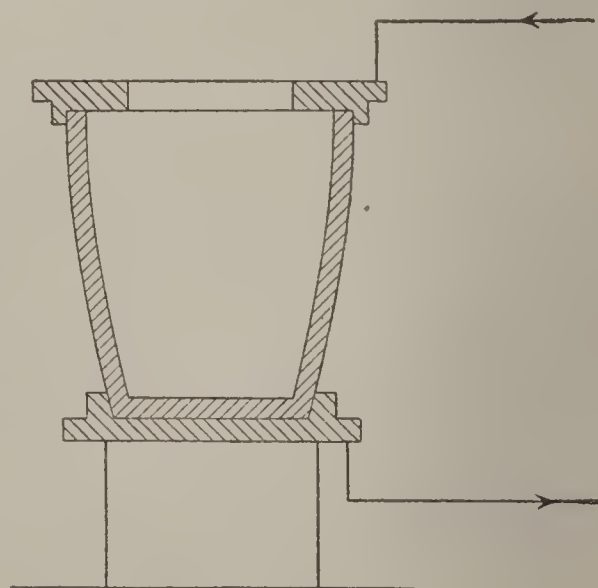


FIGURE 10.—Cross section of Helberger furnace.

An improved form of the Moissan furnace was described in Engineering^b, March 23, 1906, in which an electric current of 1,000 amperes at from 50 to 150 volts is em-

^a Stansfield, A., The electric furnace, 1907, p. 19.

^b Anon., One-thousand ampere Moissan electric furnace: Engineering, vol. 81, 1906, p. 381.

ployed. In the case of direct current this would mean 70 to 200 horsepower, and, while this is not quite as much as Moissan sometimes used, it is more than is often available for scientific experimental work. In such a furnace it is easy to produce a temperature more than double that usually obtainable by the combustion of fuel, and it is, therefore, an invaluable apparatus in the hands of the metallurgist and the chemist.

Arc heating by radiation finds its practical application in the Stassano furnace, shown in figure 5.

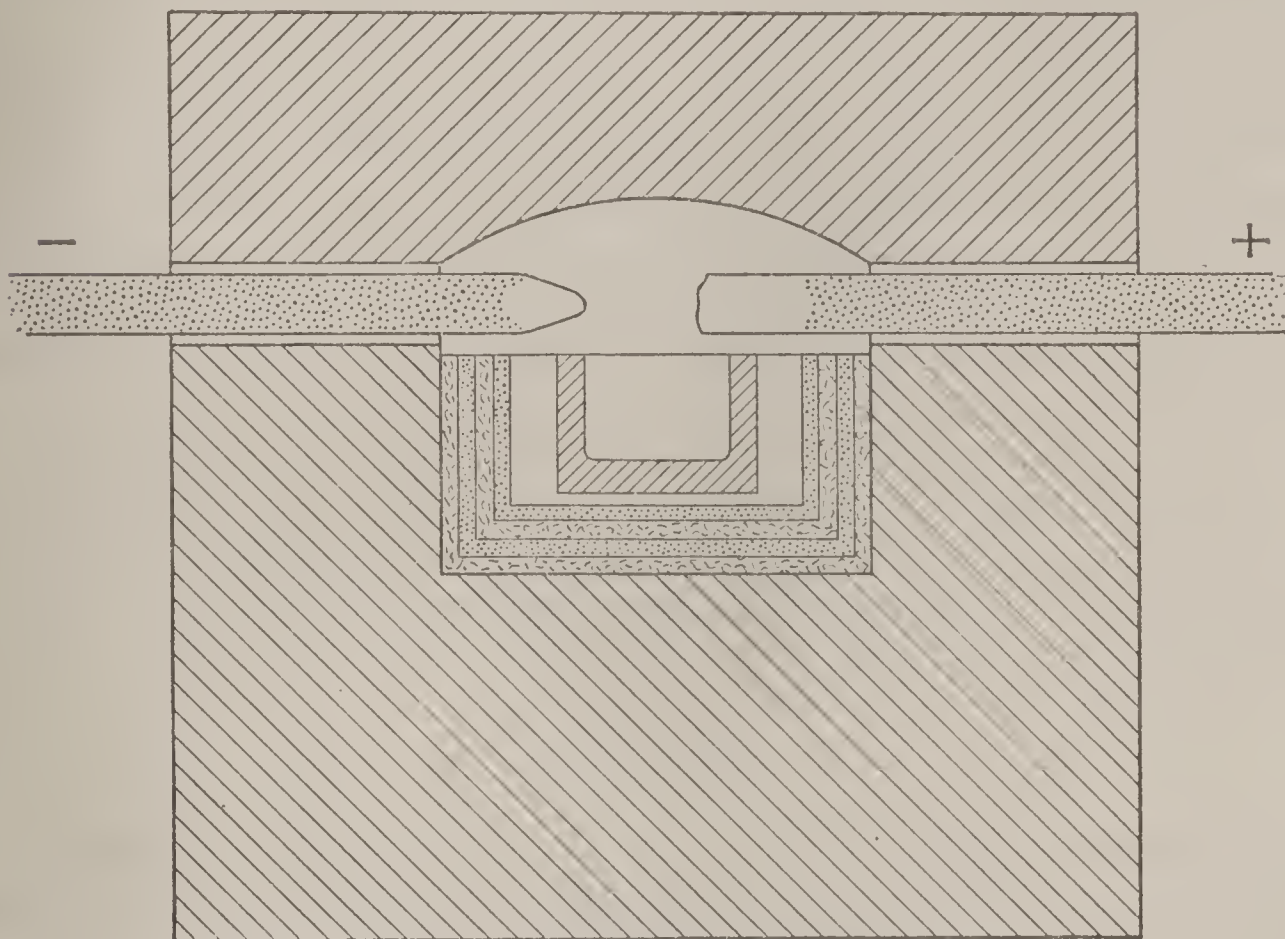


FIGURE 11.—Cross section of Moissan furnace (after Stansfield).

FACTORS AFFECTING EFFICIENCY.

As the various types of heating used in electric-furnace work have been outlined, and as the most important factors connected with the use of the electric current for this purpose have been presented, the bearing of these factors on the efficiency of the furnace will now be considered.

INDUCTION CURRENTS.

SELF-INDUCTION.

As previously stated, the strength and direction of an alternating current are continually changing—twice during each period or cycle it increases from zero to a maximum and then decreases to zero again. Owing to this fact, a conductor carrying alternating currents is surrounded by an alternating magnetic field, which, as has been shown, induces currents (electromotive force) in all conductors within

the field. As the conductor carrying the alternating current necessarily lies in the magnetic field, a current is induced within the conductor by its own field. This effect is known as self-induction, and the resulting current is known as the self-induced current.

As such a current may cause considerable loss of power under certain conditions, it is well to understand these conditions and as far as possible avoid them in the construction of electric furnaces.

An induced current is generated in every conductor that is parallel to the main current, hence care should be taken not to short-circuit the conductor carrying the induced current, since this may cause heavy losses of power. For example, in designing a furnace, an iron beam should not be placed in such a position as to cause the conductor to be short-circuited on itself. Likewise the cables and bus bars leading from the transformers to the furnace should be placed close together, for in this way there is no action on closed iron parts, or parts that lie parallel to one another, and thus the magnetic fields, as for example those made by two conductors of a single-phase current, are neutralized. These conditions are of importance only when heavy currents are employed, but heavy currents are usually used in electric-furnace work. One must remember also that currents are induced in all metal parts that are near a conductor.

EDDY CURRENTS.

The currents that are induced in the metal parts of a furnace are called eddy currents. As the metallic parts are good conductors, through which the current may be short-circuited, these parts if improperly placed may become decidedly hot and cause more or less trouble. This is especially true when the metal is magnetic, that is, a good conductor of the magnetic lines of force. For example, in furnaces in which the current is led to the electrodes through electrode holders, making these holders of cast iron or steel will cause considerable loss. For this reason the holders, which are generally water-cooled, are made of copper or brass, neither of which is magnetic.

Another method of lessening eddy-current losses is to greatly subdivide the metallic parts in which eddy currents are likely to occur, as is done, for example, in transformers and in armatures of dynamos.

HYSTERESIS.

There is also another loss from what is known as hysteresis, the lag of demagnetization. This loss occurs in magnetic conductors having considerable cross section and is due to the constant magnetizing and demagnetizing influence of the alternating current. Therefore, in furnace construction, the electrical conductors should not be surrounded by parts that are good magnetic conductors.

LAG.

If an electric current be passed through a coil of wire, the current does not immediately reach its maximum value, but does so only after a certain time has elapsed. In other words, the current lags behind the voltage, and this is known as the lag of the current.

ELECTROMECHANICAL FORCES IN THE INDUCTION FURNACE.

The nature and effect of the mechanical forces of electric origin in the bath of the induction furnace, although at present well understood, are still to many a source of apprehension in the design and operation of this type of electric furnace. These electromechanical forces, which often are of considerable magnitude, are of two kinds: (1) Attracting, due to the mutual attraction of current-carrying elements within the bath, similar to the attraction of two parallel con-

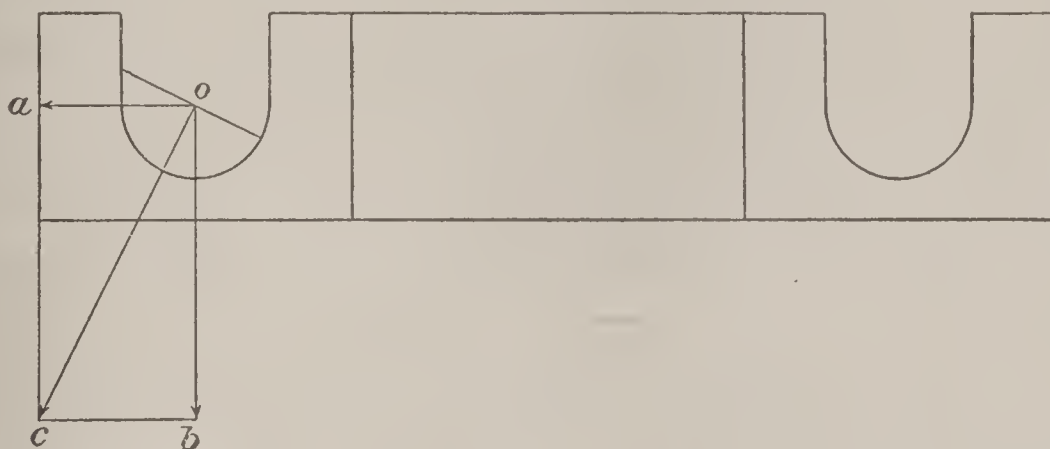


FIGURE 12.—Cross section of the bath of an induction furnace, showing effect on surface of bath of the resultant of the different forces acting.

ductors carrying current in the same direction; and (2) opposing, due to the repelling action between the bath and the primary winding, similar to the repelling action of two conductors carrying current in opposite directions.

PINCH EFFECT.

Attention was several years ago called to the first of these forces by E. F. Northrup and later by C. Hering, who observed a contraction in the cross section of the bath. The contraction was frequently so pronounced that complete rupture resulted. This phenomenon has since been generally referred to as the "pinch effect."

In figure 12 the circle represents the cross section of a circular conductor. The tendency of the electromagnetic forces of the first class mentioned is to act radially and their manifestation is a mechanical effect, the liquid particles constituting the conductor moving bodily in the direction of the forces. If, by suitable means, the pressure in the bath be measured, it is found to be greatest at the center and least at the periphery. For the pressure at the center

of a bath having a circular cross section, the following formula has developed: $P = \frac{I^2}{S}$, in which P is the pressure in dynes per square centimeter, I is the current in c. g. s. units (10 amperes), and S is the cross section of the bath in square centimeters.

Although the formula holds precisely true only for conductors of circular cross section, no great error is involved in applying it to the practical working conditions of the induction furnace.

In the bath of an annular crucible of ideally uniform cross section, this pressure would not produce the contraction known as the pinch effect, but in practice the cross section is never uniform, because of dissymmetry in the crucible, or of deflection of the plane of the crucible from the horizontal, or of the presence in the bath of unmolten non-conducting material, such as floating pieces of ore or brick. Whenever the cross section of the bath is reduced by any cause the pressure is locally increased and the result is a lateral force causing a flow to adjacent areas of lower pressure. An increase in the current will augment this effect and if the current is further increased a limit is reached, when the liquid is "squirted" out from the spot of high pressure faster than it can return, and a break in the circuit results. If the bath is mobile, the circuit will continue to close and open until freezing occurs.

The working limits imposed upon induction furnaces by the pinch effect have been studied and a formula developed, by means of which the critical current values can be calculated approximately. Experience has shown that the pinch effect does not cause difficulty in the operation of large induction furnaces, such as are used in steel making, for in these the values of the currents used are far below the critical points. The pinch effect, however, has thus far made impracticable the melting in an annular crucible of metals of low specific gravity, such as aluminum. Metals of low specific resistance, such as copper, are likewise less amenable to working in induction furnaces of the annular type, because heat generation is a function of the specific resistance, and when the specific resistance is low greater current density is required, which may exceed the critical value for the furnace.

From the standpoint of metallurgists the mechanical effects of the magnetic forces set up in the bath of the induction furnace are of value. The resulting circulation of metal in the crucible assists the heating by increasing convection and accelerates chemical action by bringing the reacting substances into more intimate mixture. Appreciation of these advantages has inspired efforts to design on the induction principle a furnace in which the danger of "pinching" will be eliminated but the desirable effect of increased circulation retained.

Such a furnace has been constructed and although still in the development stage promises speedily to find peculiar and important application in the melting of metals.

CENTRIFUGAL EFFECT.

Electromagnetic forces of the second class are set up between the bath and the primary winding. Their mechanical manifestation in the bath resembles in many respects the effect that would be produced by rotating the annular bath on its axis, and for this reason the name "centrifugal effect" has been given it.

This repelling force between the primary winding and the bath of a single-ring induction furnace is analogous to the repelling forces between the windings of a core transformer, and the theory and calculations for the latter forces may be applied to the induction furnace, as the induction furnace is in fact equivalent to a transformer with a short-circuited secondary.

In addition to the centrifugal force mentioned a second force acting upon the bath is gravity. In figure 12, representing the cross section of the bath of an induction furnace, if oa be considered to represent the centrifugal force of electromagnetic origin and ob gravity, the surface of the bath must be perpendicular to the resultant force oc . As the tendency of the pinch effect is to give the bath a circular cross section, the result of all the forces acting on the bath is to give its cross section the form shown on the left. In large furnaces operating at full load the surface of the bath is inclined about 30° to the horizontal.

There is no danger of the circuit breaking as a result of the centrifugal force on the liquid bath. When a solid starting ring breaks, however, the break is invariably due to the centrifugal force, which has exceeded the tensile strength of the ring.

The centrifugal effect, like the pinch effect, tends to eliminate inclosed gases by compression of the bath against the outer wall, and also causes circulation at right angles to the direction of the bath. In some large furnaces, such as the Frick and the Hiorth, sections of the primary winding are placed above and below the bath, thereby moderating the circulation and removing the danger of the starting ring being broken.

POWER FACTOR.

MEASUREMENT OF POWER FACTOR.

The power supplied to a furnace in watts is for a direct current the product of the number of amperes and the number of volts, whereas for an alternating current it is the product of the number of amperes and the number of volts, multiplied by a factor known as the power factor,

which varies with the construction of the furnace, as explained later. In this connection, it may be stated that alternating current is used almost entirely in furnace work, for two reasons. First, almost all large electric-power installations generate alternating current, and hence at most places only alternating current is available for furnaces, unless the current be transformed by means of a motor generator set into direct current. Moreover, a direct current is not necessary, and in fact is undesirable in furnace work, because it exercises an electrolytic action on the metals produced.

The power (watts) delivered to the furnace may be kept constant by keeping the voltage low and the amperage correspondingly high, or by keeping the voltage high and the amperage correspondingly low. If, for example, a furnace be built for 50 kw. (50,000 watts) at 50 volts, if the power factor were neglected, the ammeter would show 1,000 amperes; in other words, the number of watts divided by the number of volts gives the number of amperes. For currents of large amperage large conductors are required, thus increasing the cost of cables, transformers, electrodes, and other equipment. The use of a high voltage, although economical, is generally impractical, because most furnaces have a low electrical resistance. For example, in the Swedish and California types of electric iron-reduction furnaces the resistance is in the neighborhood of 0.01 ohm, and the voltage used varies from 40 to 80 volts with a three-phase furnace. With an average load of 500 kw. on each phase, the corresponding number of amperes per phase would be 10,000. Thus it is seen that having to handle electric currents of high amperage is one of the disadvantages of electric-furnace work of this nature.

USE OF OHM'S LAW.

As previously stated, the power used in a furnace, measured in watts, is for direct current the product of the number of amperes and the number of volts, whereas for alternating current it is the product of the number of amperes and the number of volts, multiplied by a factor known as the power factor. If the relations between the alternating and the direct current be compared, it will be found that for the direct current a definite voltage is required to force a current, I , through a resistance, R , and, according to Ohm's law:

$$E \text{ (electrical pressure)} = I \times R$$

In order to force an equal alternating current, I , through a resistance (the resistor, the charge, or whatever is used in a particular case), the fact already explained must be taken into account—that with the alternating current an electromotive force due to self-induction is generated, which is always in the opposite direction to the impressed

electromotive force. Therefore instead of the equation being $E = I \times R$, it must be written as $E_r = I \times R$, and as an additional pressure (voltage) is needed to overcome the electromotive force of self-induction, the total voltage necessary for an alternating current is $E = E_r + E_1$.

In other words, the voltage of an alternating-current voltage has two different pressure waves, and instead of the power P on the furnace being expressed by the equation $P = E \times I$, for alternating current the equation is $P = EI \cos \phi$, where ϕ is the angle of lag or lead of current, because, as has been pointed out, the effect of capacity on the circuit is to cause the electromotive force to lag behind the current. Therefore, if the current either lags or leads, the readings of the instruments used for measuring the electric current at the furnace (the voltmeter and ammeter) are not true simultaneous values, and in order to get the true values the product of the number of volts and amperes shown on the instruments must be multiplied by a factor obtained in the manner above stated.

Hence, in an alternating-current circuit the product $E \times I$ is called the apparent power, and can be measured in volt-amperes or kilovolt-amperes, a kilovolt-ampere being equal to 1,000 volt-amperes. The product $EI \times \cos \phi$ indicates the real or effective power and is measured in watts or kilowatts. Inasmuch as the determination of the cross section of an electrical conductor depends on the strength of the current the conductor is to carry, and the current strength decreases with an increasing power factor, provided the voltage and power remain unchanged, it is of course desirable to obtain the highest power factor.

As the ammeter and voltmeter on an alternating circuit do not in any way indicate what the power factor is, a special meter is used called the wattmeter, which indicates the watts or kilowatts directly. One kilowatt equals 1,000 watts.

Because of the facts stated, it is evident that in designing a furnace a high-power factor should be sought.

POWER CIRCUITS.

As previously stated, most large power circuits are three phase, which means that three lines are needed to conduct the current from the source of supply to the point where it is used, whereas with single-phase current only two lines are necessary.

In using a single-phase current the connecting of the electrodes to the source of supply is a simple matter, as may be seen by referring to figures 1, 2, 3, and others. If, on the other hand, three-phase current be used, three different types of connections may be used, as shown in figure 13.

REGULATION OF THE ELECTRIC CURRENT.

The current in an electric furnace may be regulated in one of two ways, as outlined below:

1. Inasmuch as the electrical resistance in a furnace changes with the height at which the electrode is kept above the charge or bath, the electrode may be made movable and then raised or lowered as may be necessary.

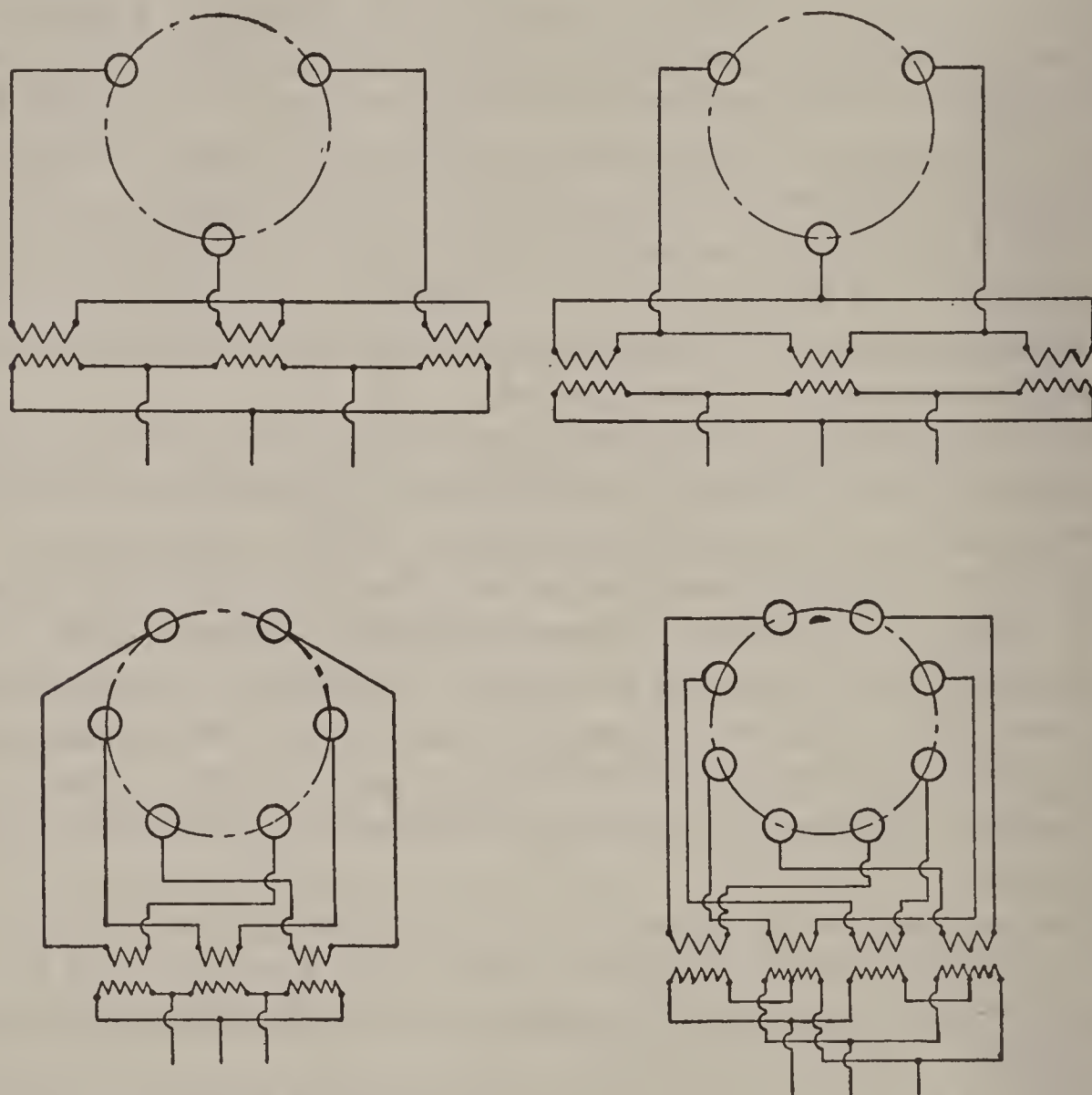


FIGURE 13.—Different types of electric furnace connections.

2. The electrode may be kept stationary and the voltage varied by means of variable voltage transformers.

The first method is employed in operating steel furnaces of the Héroult, Girod, and Electro-Metals type. The raising and lowering of the electrodes may be done by hand or by means of electric motors controlled by instruments regulated by the voltage of the furnace in such a manner as to keep the load constant.

The second method is used on the Swedish and California types of furnaces.

Van Norden ^a describes the electric control of the electric iron smelting furnaces of the Noble Electric Steel Co. at Heroult, Cal., as follows:

Electric control is through a switchboard, there being a panel for each furnace. As the current and power factor in each phase must be under observation at all times during operation, separate meters are installed in each phase. The requirements for one panel are three ammeters, three voltmeters, three wattmeters, three power factor meters, and three recording wattmeters. These are mounted across the panel in rows of three each. Under the first four sets named are three handwheels to control the voltage variation, and under these, three switches which control the entire load, and still under these are the recording wattmeters.

For operating the voltage control and the main circuit breakers there is a $7\frac{1}{2}$ -kilowatt motor-generator set, comprising a 125-volt direct-current generator, directly connected to a 10-horsepower induction motor. This set has a small panel-board mounting, a circuit breaker, ammeter, voltmeter, and two single-pole knife switches. * * * In the event that line voltage should fall, or for any other cause, the direct current supply should become deranged, there is a * * * storage-battery set, having a capacity of $7\frac{1}{2}$ kilowatts which may be instantly switched in, and thus prevent the furnace from cutting out in the case of low voltage.

ELECTROLYSIS.

Although the electrolytic or decomposing effect of the electric current finds more extended application in the electrochemical industries than it does in the production of metals from their ores, it is nevertheless more or less used for the latter purpose, especially in the production of aluminum, and for that reason is briefly discussed at this point.

ELECTROLYSIS OF WATER.

The electrolysis of water is a familiar example of the decomposing effect of the direct electric current. By the use of such a current water can be resolved into its component elements, hydrogen and oxygen. In order to do this the terminals of a direct electric current are inserted into a dish or glass containing the water. The positive terminal is known as the anode, and the negative as the cathode. When the current is turned on, it passes through the water from the positive to the negative terminal (from the anode to the cathode), and electrolyzes or decomposes the water. The hydrogen is liberated at the cathode, or negative terminal, which may be designated the negative electrode, and the oxygen is liberated at the anode or positive electrode.

ELECTROLYSIS OF COPPER SULPHATE.

Similarly, if a direct current be passed through a solution of copper sulphate, which, in such case, is termed the electrolyte, copper will be deposited at the cathode and sulphuric acid gas will be given off at the anode. In refining cast copper, the anode is made of the

^a Van Norden, R. W., Jour. Elect. Power and Gas, vol. 29, 1913, p. 1.

impure metal, and a solution of copper sulphate is used as the electrolyte. Under the action of the electric current, the copper anode dissolves, and pure copper is redeposited at the cathode. Any substances other than copper in the anode, are, if the current is properly regulated, liberated and collect on the bottom of the tank containing the electrolyte.

PREREQUISITES OF AN ELECTROLYTE.

In electrolyzing a solution of copper sulphate, the water, which is a more stable combination than copper sulphate, is not broken up, as the current acts on the least stable compound. On the other hand a solution of aluminum sulphate is more stable than water, and hence an electric current would decompose the water and not the aluminum sulphate. Such being the case, the following prerequisites must be observed in preparing an electrolyte, namely: For the substance to be electrolyzed a solvent that is more stable than the salt, or the pure salt in a state of fusion, must be employed. Aqueous solutions must be avoided when the water can react with the metal produced. For example, an aqueous solution can not be used in the production of metallic sodium, because water reacts with sodium, forming caustic soda and hydrogen.

ESSENTIAL DETAILS OF ELECTROLYSIS.

In electrolytic work, the quantity of metal produced depends chiefly on the quantity and voltage (electrical pressure) of the current that is passed through the electrolyte, thus bringing about a definite amount of decomposition which is always the same for the same solution. The quantity of metal that will be liberated by electrolysis per unit of time by a given current may be calculated in the following manner: If a current of 1 ampere be passed through acidulated water for a period of one second, 0.0104 milligram of hydrogen will be liberated. Therefore, the quantity of metal obtainable by passing a like current through a copper sulphate solution may be determined from the equation $\frac{C \times W}{V} = x$, in which C represents the constant 0.0104, W the atomic weight of the metal, and V the valency of the metal in the particular solution. However, the quantity of metal shown by this calculation is not always obtained, owing to the secondary reactions that take place, such as the liberation of hydrogen instead of the metal, the redissolving of metal in the electrolyte, and the leakage of the current. For this reason, what is termed current efficiency, by which is meant that proportion of the current that is effective in liberating the metal, has to be taken into consideration.

In order to pass a stated quantity of electrical energy through a given solution in a stated time, the electrical current must be kept at a definite voltage, or pressure, for that particular solution. Below this pressure, or voltage, electrolysis will not take place.

Inasmuch as the current in passing through the electrolyte meets with resistance, heat is generated, the quantity of which is proportional to the square of the current and to the resistance of the electrolytic cell. As this generation of heat represents the expenditure of electrical energy, the higher the temperature, the less efficient is the decomposing effect of the current. However, in the electrolysis of fused salts or compounds, the heat generated through the resistance of the electrolyte to the passage of the current serves to keep the electrolyte fused and so is of use instead of being wasted.

The nature and the composition of the anode are of great importance in electrolytic work. If possible the anode should meet the following seemingly contradictory requirements: The anode material should be insoluble in whatever is liberated in contact with it as the result of the electrolytic action of the current, but if the material is soluble, it should be inexpensive, for it will be dissolved in proportion as the metal is deposited. In the manufacture of aluminum, the carbon electrodes (the anodes) are consumed by the oxygen liberated by the decomposition of the alumina of the electrolyte into aluminum and oxygen, and the weight of electrodes consumed about equals the weight of aluminum produced. On the other hand, as regards the amount of energy needed to break up a given weight of compound into its elements, more voltage is required for an anode that does not dissolve than for one that does.

In electric-furnace work, electrolysis is involved in reducing metals from ores when the metal is present in the form of a salt or a mixture of salts which must be fused and the metal deposited from the fused mixture by a direct electric current.

MECHANICAL CONSTRUCTION OF FURNACES.

The electrical side of the subject has been briefly discussed in the foregoing pages. The factors that must be taken into consideration in the actual construction of an electric furnace will now be noted.

FACTORS AFFECTING OUTPUT.

Naturally, in the construction of a furnace, the first matter to be taken into consideration is output. Inasmuch as a small furnace, using a comparatively large amount of power, may produce as much as a larger furnace using a comparatively small amount of power, the nature of the process must be considered in deciding what size of furnace will be employed. Then, too, one must remember that large electric furnaces, large as compared with modern copper and iron blast furnaces, have not been successfully operated as yet, and hence if the desired production is, say, 100 tons or more of metal a day, the size of the unit to be employed will first have to be determined. As regards this point, the facts that should be borne in mind are as follows: (*a*) The relative loss of heat through the walls is less for a large than for a small furnace, and therefore each unit should

be as large as may be feasible; but (b) in order to reduce the total heat loss the furnace should be as small as possible for the stated output.

In other words, for a given output a smaller furnace is more efficient than a larger furnace, but, as before stated, the nature of the process must be taken into consideration in determining relative economy. As a comparatively large amount of energy is required to keep the temperature of the furnace uniform, it is of course better to employ continuous processes wherever feasible, or to leave as little time as possible between charges if an intermittent process be used.

CALCULATION OF ENERGY REQUIRED.

When the output of the furnace has been determined, the amount of energy that will be required to give this output has to be calculated. In this calculation the following factors have to be taken into consideration: (a) The heat available from such chemical reactions during the process as give up heat, that is, as are exothermic. Illustrations of such reactions are the burning of carbon to carbon monoxide, or of carbon monoxide to carbon dioxide, or the formation of a silicate. The other factor is (b) the heat that is absorbed during the process. The absorbed heat includes that required to raise the raw material to a smelting temperature, or to a temperature at which the reactions necessary to the process take place, and also the energy that is absorbed in such reactions, as for example, the reduction of iron oxide to iron, of silica to silicon, or of calcium from calcium oxide. Almost all of these changes involve the specific heat, generally the latent heat of liquefaction, and sometimes, too, that of vaporization. Also, allowance must be made for loss of heat by conduction through the furnace walls and through the electrodes. These losses are discussed in subsequent pages.

As a result of these calculations a heat balance sheet of the process may be constructed. The following example^a will serve to illustrate this point. Let it be assumed that an iron ore is to be smelted and that the calculations for determining the amount of heat available and necessary show that for each ton of metal to be produced 1,053,532 calories must be supplied electrically and that the desired output is 1 ton of metal per hour, then each ton will require $1,053,532 \div 3,600 = 292$ calories per second. The total horsepower required to furnish this heat, $\frac{292 \times 425 \text{ kg. m.}}{75} = 1,654$ horsepower.

Which means that theoretically there would have to be supplied to the furnace a constant current of 1,654 horsepower in order to produce 1 ton of pig iron per hour.

^a For further information on the subject of calculations of this kind see Richards, J. W., *Metallurgical calculations*, 1907, pt. 2, p. 403; and Yngstrom, Lars, *The electric production of iron from iron ore*: Engineer (London), Feb. 26, 1900, p. 206.

CURRENT AND VOLTAGE NECESSARY.

After the amount of current necessary to carry on the process has been ascertained, the voltage at which the current is to be supplied may be determined. The choice of voltage is governed largely by the construction of the furnace. Theoretically it is best to have the voltage as high as possible, but practical considerations generally prohibit the use of a high voltage, and hence a comparatively low voltage is used in electric-furnace work. For example, the voltage used for an iron-reduction furnace ranges from 40 to 80 volts, 50 volts perhaps being the average. In the case of the Héroult steel furnace 90 volts is used for the three-phase furnace at South Chicago, whereas for a Héroult single-phase furnace a voltage of 100 to 110 volts is ordinarily used. In an induction furnace the primary voltage may be as high as 3,000, whereas in the Moissan arc furnace it is about 110. As a rule, in arc furnaces the arc should be as long as other considerations permit.

PROPER SIZE OF INTERIOR OF FURNACE.

Of course waste space in the interior of a furnace greatly diminishes efficiency. On the other hand, it is sometimes necessary to allow for what may seemingly be waste. For example, in the Swedish electric iron-reduction furnace the roof of the crucible is so constructed as to allow a gas space between the surface of the charge and the roof. In this way the roof, at the point where the electrodes enter it, is somewhat protected from the heat developed in the charge. Or it may be found best to allow part of the material forming the charge to chill along the walls of the furnace in order to protect the walls. In such instances, however, the material must be considered as a part of the wall of the furnace and not an available part of the interior of the furnace. As a rule the interior of the crucible of the furnace should be made as small as practicable.

The interior dimensions of some furnaces in actual practice are given below:

Dimensions of three furnaces.

Type of furnace.	Location.	Prod-uct.	Ca-pac-ity.	Power.	Internal dimensions.			Thickness of lining.			Diam-eter of elec-trodes.
					Length.	Width.	Depth.	Ends.	Sides.	Roof.	
			Tons.	Kilowatts.	Ft. in.	Ft. in.	In.	In.	In.	In.	In.
Héroult.	Braintree, England.	}Steel..	2.5	300	{ a 4 2	1 8	31	14	12	12	14
Do..	South Chicago, Ill.	...do..	15	1,200-1,500	{ b 5 2	2 7		18	12	14
					(c)	(c)	d 36				
Girod...	Ugine, France.	}...do..	2.5-3	300	{ a 3	3	31
					{ b 6	6					

a Bottom. b Top. c Circular; diameter, 120 inches. d Approximate.

PROPER SHAPE OF CRUCIBLE.

The next item to be taken into consideration after the cubic contents of the crucible has been determined is the shape of the crucible. Theoretically, the spherical form is best so far as heat losses and general efficiency are concerned, but here, as in the determination of all points that have to be considered in the construction of a furnace, the special requirements to be met rather than the theoretical considerations will have the most weight.

TYPES OF FURNACE LININGS.

The kind of refractories that will be used in the lining of the furnace will depend on the nature of the product—that is, whether it is to be a solid or a liquid, and whether it will be acid, basic, or neutral in its chemical action.

ACID REFRACTORIES.

Silica.—Silica sand, either as such or made into brick, is most commonly used when a siliceous refractory material is desired. A typical analysis of a silica sand is as follows:

Typical analysis of silica sand.

	Per cent.
Water.....	0.24
Silica.....	97.25
Alumina and iron oxide.....	.16
Lime.....	.08
Alkalies.....	.36
Magnesia.....	.39
Loss on ignition.....	.36

When the sand is made into bricks a small percentage of lime or some other similar substance is used as a binding material. By reason of the binder the melting point of the bricks is lower than that of the pure sand, and bricks made of pure fused quartz would be more refractory. There is a possibility that such bricks may soon be available for use in furnace linings. A good silica brick should not contain less than 95 per cent of silica.

Dinas brick.—The name Dinas was applied originally to a brick made in Wales, from a pure sandstone that contained as much as 98.3 per cent and not less than 96 per cent silica. A Dinas brick that contains 98 per cent silica contains about 0.72 per cent alumina (Al_2O_3) and 0.10 to 0.20 per cent each of iron oxide (Fe_2O_3), lime (CaO), and alkalies.

Although Dinas bricks are capable of withstanding high temperatures, they have a coarse texture, are rather friable and brittle, and do not withstand abrasion. Two grades of these bricks are used in the

United States, one being imported and the other made in domestic plants. An analysis ^a of an American brick is as follows:

Analysis of an American Dinas brick.

	Per cent.
SiO ₂	94.07
Fe ₂ O ₃ and Al ₂ O ₃	3.66
CaO.....	1.39
MgO.....	.19

The Dinas bricks differ from ordinary silica bricks in the binder used. In many silica bricks the binding material is clay, whereas in the Dinas bricks, as before mentioned, the binding material is a lime silicate.

Ganister.—The term “ganister” is used loosely. According to Fulton ^b it means a siliceous material of about the following composition: Quartz, 83 per cent; clay, 13 per cent; and impurities and moisture, 4 per cent. Any siliceous material containing silica and clay in the above proportions may be used as ganister, provided it does not contain fluxing ingredients enough to lower materially the refractoriness of the product. Ganister can be made by mixing a highly siliceous material with enough clay to act as a binding material. It is not used as bricks, but as loose material, which is tamped into place as in furnace bottoms.

Fire clay.—Fire clay is sometimes classed as an acid refractory and sometimes as a neutral one, owing to the fact that in different clays the proportions of siliceous and aluminous contents, which together should constitute 92 to 98 per cent of the clay, differ widely. The refractoriness of a clay depends upon the proportion of those constituents which tend to lower the melting point of the clay. For the most part these constituents are ferric oxide, titanium oxide, lime, magnesia, and oxides of sodium and potassium. Of these constituents, ferric oxide probably lowers the refractoriness of the clay more than any other, especially if the clay also contains carbonaceous matter. Owing to the presence of the mineral, kaolinite fire clays are plastic when mixed with a comparatively small proportion of water, and for this reason are easily molded into bricks of any required shape. In practice it has been found that the texture of a brick governs to a certain degree its refractoriness. For instance, a brick made of a fine-grained clay is not as refractory as one made from a clay in which there is a large proportion of coarse particles, even if the two clays have the same chemical composition. The refractoriness of a fire brick is much increased by the coarseness of texture; that is, if a fire brick be composed of a ground mass of fine grains in which are embedded comparatively coarse grains, the greater the per-

^a Fulton, C. H., Principles of metallurgy, 1910, p. 334.

^b Fulton, C. H., op. cit., p. 335.

centage of coarse grains the more refractory will be the brick. Fulton ^a has shown that the principle involved is the same as that governing the formation temperature of slags.

BASIC MATERIALS.

Lime.—One of the most abundant basic refractory materials, and theoretically one of the best, is lime (CaO), which has a melting point of about 1,900° C. Practically, however, pure lime is of no use as furnace lining because of the rapidity with which it slakes when exposed to air. It has been found,^b however, that if the lime be fused, it will resist moist air for several days, whereas ordinarily it will do so for only a few hours.

Magnesia.—Probably the most important of the basic refractory materials is magnesia (MgO), which, in electric furnace work, is used mostly in the form of brick. The bricks are made by mixing 90 per cent of thoroughly burned magnesite with 10 per cent of incompletely burned material, which is added for the purpose of causing the bricks to set. The mixture is then molded into shapes and burned.

The bricks are dense and hard, and can be heated up to 1,800° or 1,900° C. The objections to the use of magnesia bricks are as follows:

1. They are good conductors of heat, and when hot become fairly good conductors of electricity.

2. They expand somewhat on heating and spall or flake off when subjected to sudden changes of temperature.

3. They are liable to go to pieces if subjected to a heavy load when hot.

4. They are liable to shrink and crack when heated to high temperatures.

Electrically calcined magnesite.—If the magnesite be calcined in an electric furnace, where it is subjected to a much higher temperature than is the ordinary calcined magnesite, it is less liable to shrink and crack, and does not absorb carbon dioxide gas as readily as does ordinary calcined magnesite. Fitzgerald ^c states that several tons of magnesia have been electrically calcined and tested in a great variety of furnaces, and in every case excellent results have been obtained. He also states that a resistance furnace has been developed that will calcine the magnesite to the desired degree.

Bauxite.—Bauxite is used for making both basic and neutral refractory linings, the nature of the chemical action depending largely upon the composition of the bauxite used. As compared with magnesia, bauxite, which is extensively mined as an ore of aluminum, is cheaper. It can be made into a very hard and tough brick, the tensile

^a Fulton, C. H., Principles of metallurgy, 1910, p. 272.

^b Fitzgerald, F. A. J., Refractories: Met. and Chem. Eng., vol. 10, 1912, p. 129.

^c Fitzgerald, F. A. J., Loc. cit.

strength of which may be as much as 10,000 pounds per square inch. When used for making brick, bauxite has to be selected with care, as it must contain enough silica and iron oxide to make a firm brick through the combination and incipient fusion of the two oxides. On the other hand, it must not contain too much silica, as that lowers its refractoriness. If the brick is to be used as a basic refractory material, the silica content should not exceed 12 per cent.^a The composition of a bauxite suitable for brickmaking is approximately as follows:^b

Approximate analysis of bauxite for making brick.

	Per cent.
Silica (SiO_2).....	4 to 7
Iron peroxide (Fe_2O_3).....	3 to 5
Alumina (Al_2O_3).....	60
Water (H_2O).....	30

Although bauxite is very refractory, it must be almost completely calcined, a difficult attainment, before it can be used for furnace lining or else it will shrink excessively at furnace temperatures. Calcination causes it to lose its combined water and thus destroys its plasticity. Because of these disadvantages it is seldom used in electric furnaces. At Braintree, England, the Lakes & Elliott Foundry Co. have used bauxite bricks in the construction of the roof of their Héroult steel furnace. The roof lasts from 14 to 28 heats. The short life of the roof, as compared to the life of the roof of other furnaces, is doubtless due to the fact that the construction of the furnace was such as to prevent the usual amount of radiation.

NEUTRAL LININGS.

Carbon.—Carbon is neutral and is one of the most highly refractory substances known. It is generally used in the form of lamp black or retort carbon.

Although carbon is capable of withstanding the action of slags, it has the drawback of being readily attacked by metals which have a strong affinity for it. For this reason carbon in the form of brick, or rammed with tar, has never come into practical use in open-hearth furnace bottoms, but the bottoms of furnaces used for making ferro-alloys, such as ferrochrome and calcium carbide, are nearly all made of carbon. Moreover, these bottoms are as durable as any other part of the furnace, in fact more so. At the works of the Keller-Leleux Co., Livet, France, the furnaces, when once charged and started, are run continuously for an average period of a couple of years. The company states that the period may be made as long as four years.^c Also, when, in 1912, one of the authors of this bul-

^a Fulton, C. H., Principles of metallurgy, 1910, p. 340.

^b (Editorial), Met. and Chem. Eng., vol. 8, 1910, p. 107.

^c Copeman, S. M., Bennett, S. R., and Hahe, H. W., Manufacture and transport of ferrosilicon: Met. and Chem. Eng., vol. 8, March, 1910, p. 135.

letin visited the plant of the Meraker Electric Smelting Co., Kop-peraaen, Norway, he was informed that one of its furnaces, which have carbon bottoms and are used in the production of ferrochrome and calcium carbide, had been running continuously for 22 months and the other for 6 months. That the carbon bottoms in these furnaces are not destroyed is probably because enough carbon is added to the charges to satisfy the alloys produced and because the bottoms are not exposed to oxidation.

The bottoms are made of lampblack or retort carbon, which is mixed with tar, heated, and then rammed into place.

Chromite.—Chromite, usually termed chrome iron ore, is a sesquioxide of chromium. A typical analysis of chromite is as follows:

Typical analysis of chromite.

	Per cent.
Sesquioxide of chromium (Cr_2O_3).....	38 to 40
Alumina (Al_2O_3).....	24.5
Iron peroxide (Fe_2O_3).....	17.5
Silica (SiO_2).....	3.25
Magnesia (MgO).....	15.0

Although chromite is extremely infusible, this property has the disadvantage of making its thorough sintering difficult. For this reason, chromite, although not affected by comparatively high temperatures, will not withstand mechanical erosion, and hence is not suited to furnace bottoms. It is well suited to the construction of furnace walls above the slag line, where they are not subjected to mechanical erosion of any sort.

Alundum.—Alundum is made by purifying and fusing bauxite in an electric furnace. Two forms, a white and a brown, the former being the purer, are on the market. The melting point of white alundum is between $2,050^\circ$ and $2,100^\circ$ C. and that of the brown product is not more than 50° lower.^a

Alundum has been formed into bricks which were used for making the roof of an electric furnace. In this same furnace a roof built of silica brick was destroyed in 5 or 6 hours, whereas the alundum roof, at similar temperatures, lasted 40 or 50 hours. However, as pointed out by Fitzgerald,^b although alundum bricks are capable of withstanding high temperatures when used in the construction of the roof of an electric steel furnace, they are not entirely suited for such construction for the following reasons:

1. The lime vapors that rise from the intensely heated basic slag in the furnace react with the alundum brick and in time destroy the roof.

^a Fitzgerald, F. A. J., *Refractories: Met. and Chem. Eng.*, vol. 10, March, 1912, p. 129.

^b Fitzgerald, F. A. J., *Idem.*

2. Although alundum bricks outlast silica bricks, yet because of their extra cost their use in electric-furnace construction is not warranted.

Carborundum.—When sand, sawdust, and coal are heated together in an electric furnace at a high temperature, a carbide of silicon is formed which is known as carborundum. This carborundum may be crushed and made into brick by mixing the crushed product with a suitable binder, such as tar, glue, or sodium silicate. The fusing point of carborundum itself is about $2,200^{\circ}\text{C}$. Three samples of carborundum brick were recently submitted to the Bureau of Standards for examination, and the melting point of all three proved to be about $1,720^{\circ}\text{C}$. The failure of these bricks was due to the destruction of the binder, not to the fusion of the material as a whole. After the binder is destroyed, the remaining material has too little strength to last as a brick, often crumbling from its own weight. The binding material is largely vaporized, perhaps after a chemical reaction with the carborundum, as is shown by a considerable loss in weight and by the remaining material containing practically nothing but carborundum. All three samples behaved in practically the same way under test. When they were heated in a vacuum, loss of the binding material became noticeable at about $1,725^{\circ}\text{C}$. and became rapid at $1,750^{\circ}\text{C}$. When the samples were heated at atmospheric pressure, loss occurred at those temperatures, but it did not become rapid until a higher temperature was reached. At $1,860^{\circ}\text{C}$. considerable changes occurred in half an hour.

Crystolon.—Crystolon is a trade name given to a crystalline form of carborundum made by the Norton Co. Being crystalline, it is made at a higher temperature than the amorphous form and is more refractory. The Norton Co. and the Fitzgerald & Bennie laboratories have done a great deal of experimenting with crystolon as a refractory material. Fitzgerald ^a describes the results of the experiments as follows:

Under ordinary conditions crystolon is infusible, but when heated to a sufficiently high temperature in a reducing or neutral atmosphere it is decomposed, silicon vaporizing and the carbon remaining behind as graphite. If heated to a sufficiently high temperature in a strongly oxidizing atmosphere, slow oxidation occurs, the carbon burning to carbon monoxide and the silicon forming silica. The latter appears to form a protective coating which prevents further oxidation, or at least prevents it from proceeding rapidly.

Fitzgerald ^b has patented a method for making crystolon into any desired form without the use of a binding agent. He mixes finely pulverized silicon carbide with some temporary binder, such as a solution of glue or dextrine, molds the mixture into the desired form, and then heats it in an electric furnace to the temperature at which

^a Fitzgerald, F. A. J., *Refractories: Met. and Chem. Eng.*, vol. 10, 1912, p. 129.

^b Fitzgerald, F. A. J., U. S. patents 650,234 and 650,235.

silicon carbide is formed. This causes a crystallization or recrystallization of the silicon carbide, and a strong, very refractory article is obtained.

As one of the greatest drawbacks to the use of the electric furnace in metallurgy has been the difficulty of maintaining the roof of the crucible, the possibility of using crystolon bricks in roofs is most important, and it is to be hoped that the material will prove satisfactory for this purpose.

SUITABLE REFRACTORIES.

The selection of a suitable refractory for electric furnaces is often difficult. After decision has been reached as to whether an acid, basic, or neutral refractory is desired, the next step is to determine which may be the best refractory under the conditions to be met.

Owing to defects in the process of manufacture, the melting point of a brick is not always a measure of its refractoriness. For example, the use of fire-clay brick as a refractory may be contemplated, and two samples may be submitted for examination. One is found to have a lower melting point than the other, and so it may be said to be less refractory, but in service the one with the lower fusion point may be found to last three to five times as long as the one with the higher fusion point. If the structure of two such bricks be examined, the more durable brick will probably show a dense stony-like structure, in which the component grains are firmly held by the bond, whereas the one with the higher fusion point will probably show a crumbly fracture. By reason of these structural differences in the bricks the one with the lower fusion point wears away slowly, whereas the other, as soon as its surface is worn off, falls to pieces. The composition of the latter brick is correct, but the brick was improperly made. For this reason, in the selection of a fire brick its analysis and fusion point are not reliable indices of its refractoriness. As Rigg^a has pointed out, the properties possessed by a product that may be properly regarded as refractory may include any or all of the following, according to the conditions of service:

1. Absolute infusibility at the highest working temperature.
2. Complete absence of deformation and shrinkage under working conditions.
3. Mechanical strength.
4. Complete resistance to penetration of vapors, slags, etc.
5. A chemical composition fitted to withstand as completely as possible the corrosive action of the substances to which the bricks are exposed.
6. Equality and fixity of form and dimensions.

^a Rigg, Gilbert, Defects in refractory brick and their causes: *Met. and Chem. Eng.*, vol. 8, May, 1910, p. 237.

As all of these properties depend on the care with which the bricks are manufactured, as well as on their composition, there is obviously a good reason for purchasing bricks from firms whose reputation is well established and who are able, by reason of their experience, to turn out a product that will meet the requirements mentioned by Rigg.

It is well to remember that the structure of a brick—that is, whether it is porous or not—has a great deal to do with its fusion point, if it is to be exposed to furnace gases or slags. Especially is this true of slags; for, obviously, if the brick be porous, the slag will penetrate the pores of the brick and will likely act as a flux for its constituents. Rigg cites from his own experience an example in point. The melting point of a fire-clay mass exposed to a penetrating slag fell from over 1,700° C. to 1,400° C., although, except for a coat of glaze, the mass appeared unchanged. Under the microscope the filling of the pores by slag was plainly visible.

MELTING POINTS OF REFRACTORY MATERIALS.

The following table shows the melting points of some of the most important refractory materials used in furnace construction:

Melting points of important refractory materials.

Name of substance.	Melting point.		Chemical nature.
	° C.	° F.	
Fire clay.....	1,400-1,800	2,250-3,300	Neutral
Silica brick.....	1,700-1,800	3,092-3,272	Acid
Silica, pure.....	1,830	3,326	Do.
Lime.....	2,040	3,704	Basic
Magnesia brick.....	2,000	3,632	Do.
Alumina.....	2,100	3,844	Neutral
Carborundum.....	2,220	4,060	Do.
Carbon.....	^a 3,700	^a 6,756	Do.

^a Boils.

In Technologic Paper 10^a of the Bureau of Standards the melting points of various bricks are given, as follows:

Melting points of various bricks.

Kind of brick.	Melting point.	
	° C.	° F.
Fire clay.....	1,555-1,725	2,863-3,169
Bauxite.....	1,565-1,785	2,881-3,277
Silica.....	1,700-1,705	3,092-3,101
Chromite.....	2,050	3,754
Magnesia.....	2,165	3,993

^a Kanolt, C. W., Melting points of fire bricks: Tech. Paper 10, Bureau of Standards, 1912, p. 16.

As pointed out in the paper by Kanolt above mentioned, "it does not seem possible to establish any very definite relation of a fire brick to its melting point. Since the bricks contain about eight different constituents in quantities sufficient to affect the melting point, and since the melting point may also be affected by lack of homogeneity in the material, it is obvious that a prediction of the melting point upon the basis of a chemical analysis would be uncertain."

In brief, it may be said that in the construction of an electric furnace the choice of the refractory material depends on its capability to withstand fusion in the walls and the crucible of the furnace when subjected to the action of the solids, liquids, and gases that are employed in connection with or are produced at the working temperature of the process.

PROPER THICKNESS OF FURNACE WALLS.

When the kind of refractory material that will be best suited to the furnace has been determined, the next item is the most desirable thickness of the walls of the furnace, especially of the crucible. This detail is of much importance because the efficiency of an electric furnace depends largely on the proportion of the heat value of the electrical energy that is utilized in heating the contents of the furnace. This matter is especially important in the electric furnace, because the electric energy used in heating the furnace is generally much more expensive than are the fuels similarly used. On the other hand, as has been pointed out, the heat supplied to a combustion furnace is largely carried away by the escaping flue gases, whereas in the electric furnace this loss does not occur. Hence in the construction of the electric furnace the main considerations are the heat conductivities of the furnace materials and the ratio of heat loss per unit of volume.

Naturally, the heat loss per unit of volume depends largely on the heat conductivity of the materials that are used in the construction of the furnace.

HEAT CONDUCTIVITIES OF FURNACE-WALL MATERIALS.

In the tabulation following are given the heat conductivities of various refractory materials.^a

^a Stansfield, A., *The electric furnace*, 1907, p. 57.

Heat conductivities of furnace materials.

[In C. G. S. units.]

Material.	Temperature.	Conductivity.
	° C.	
Fire-clay bricks.....	0- 500	0.00140
....do.....	0-1,300	.00310
Alumina bricks.....	0- 700	.00204
Magnesia bricks.....	0-1,300	.00620
Lime.....	20- 98	.00029
Carborundum sand.....	18- 98	.00050
Quartz sand.....	18- 98	.00060
Fire-brick dust.....	20- 98	.00028
Infusorial earth.....	17- 98	.00013
....do.....	0- 650	.00038

In connection with the conductivity figures Stansfield ^a remarks as follows:

The figures indicate the number of gram-calories of heat that would pass in one second through a centimeter cube of the material if the hot and cold sides differed in temperature by 1 degree centigrade. The best conductivity varies with the temperature, being greater at high temperatures, as is shown by the first two and the last two items in the list. The figures represent the mean conductivity for the range of temperature indicated, and were probably obtained by measuring the flow of heat through a wall of definite thickness and area, the two sides of which were maintained at the temperatures mentioned. It will be understood that a material having a high conductivity for heat, as shown in the table, would, if used in the construction of a furnace wall, allow a considerable amount of heat to escape and be wasted.

PROPER VOLUME OF CRUCIBLE.

After the proper refractory material for use in the construction of a furnace has been selected, it is necessary to determine whether the laboratory or crucible of the furnace has the proper volume, for the latter is largely governed by the heat conductivity of the materials used in the construction of the crucible, and likewise the degree of heat insulation possible or feasible. On the other hand, it may be necessary to provide for carrying away heat from some parts of the furnace that may be exposed to corrosive slags or to a very high temperature. In order to bring this about, air cooling or water cooling may be adopted.

INSULATION DESIRABLE.

If it be desirable to insulate the walls of the furnace, the degree of insulation desirable must be determined. To effect perfect insulation would be, as pointed out by Hering,^b to surround the furnace walls with a material that at each point has the same temperature as that of the adjacent point of the conductor. In attempting to obtain perfect insulation, it is quite possible to overinsulate, and thus cause serious damage to the inner walls of the crucible; that is, in

^a Stansfield, A., loc. cit.

^b Hering, C., The thermal insulation of furnace walls: Met. and Chem. Eng., vol. 10, February, 1912, p. 97.

certain instances, in order to maintain the inner walls it may be necessary to permit a considerable flow of heat from the inner walls through the outer walls in order to prevent the destruction of the inner walls. The degree of insulation that should be attempted can be determined only by careful calculation and experimentation. The following table of insulating values is given by Snyder.^a

Thermal conductivity of refractories.

D—Degrees centigrade required to force a heat flow of 1 kilowatt per square foot through 1 inch of thickness of the material. T—Temperature at hot side of material. t—Temperature at cold side of material.

	t	T	D
Alumina brick.....	0	700	320
Ashes, wood.....	0	100	3,840
Brick dust, coarse.....	0	100	1,680
Carbon, retort, solid.....	0	100	45
Carbon, electrode.....	100	400	7.1
Carbon, electrode.....	100	800	5.2
Carbon, electrode.....	100	1,200	5.0
Carbon, electrode.....	100	1,600	4.5
Carborundum sand.....	18	98	1,300
Cement.....	0	700	3,800
Charcoal.....	0	100	2,960
Clinker, small grains.....	0	700	600
Coke, powdered.....	0	100	1,480
Copper, metallic.....	0	30	0.7
Fire brick.....	0	500	465
Fire brick.....	0	1,000	155
Fire brick.....	0	1,300	105
Fire-brick dust.....	20	98	2,300
Graphite, dust.....	20	100	1,630
Graphite, electrode.....	100	400	1.9
Graphite, electrode.....	100	800	2.1
Graphite, electrode.....	100	1,200	2.3
Graphite, electrode.....	100	1,600	2.5
Infusorial earth.....	17	98	5,000
Infusorial earth.....	0	650	1,720
Iron, metallic.....	100	200	4.4
Lime.....	20	98	2,250
Magnesia, brick, dust.....	20	100	1,300
Magnesia, brick.....	0	1,000	92
Magnesia, calcined, Grecian, granular.....	20	100	1,450
Magnesia, calcined, Styrian.....	20	100	1,920
Magnesia, calcined, light, porous.....	20	100	4,100
Quartz sand.....	18	98	1,090
Silica brick, burned to 1,050° C.....	0	1,000	330
Silica brick, burned to 1,310° C.....	0	1,000	210

As has been stated, the ratio of heat loss per unit of volume will be inversely proportional to the dimensions of the furnace when the walls are of equal thickness in each furnace considered. However,

^a Snyder, F. T., Flow of heat through furnace walls: Trans. Am. Electrochem. Soc., vol. 18, 1910, p. 243.

as a matter of fact, smaller furnaces generally have thinner walls than do the larger ones, and so, in addition to the heat loss that may be calculated as being inversely proportional to the dimensions of the furnace, the loss of heat through correspondingly thinner walls must be considered. For this reason a large furnace is more efficient than a small furnace; that is, the amount of work performed per unit of electrical energy expended is greater in the large furnace than it is in the small one. As Stansfield^a has pointed out, "in the extreme case of a small furnace constructed as an exact model on a scale of 1 inch to the foot of a large furnace, the heat loss for each cubic inch of the model would be 144 times as great as from the large furnace, provided, of course, that both attained the same temperature. In other words, if the furnaces were merely being kept hot, no work being done in them, the small furnace would need 144 times as much heat per cubic inch as the larger furnace in order to keep it heated to the same temperature."

In some electric-furnace processes, as making carborundum by the Acheson furnace, it is possible to develop the heat within the charge being heated and to interpose between the hot inner part of the charge and the exterior walls of the furnace enough charge to prevent the exterior walls from being heated. Generally such processes are not continuous and the outside of the charge is not attacked. Naturally the ideal condition in electric-furnace work is to have the process continuous, with the cold charge advancing from the cold exterior part of the furnace to the hot interior part. In this way, of course, the heat radiating from the center of the furnace is absorbed by the cold charge advancing toward the center and is not lost, as it is when it is dissipated through the furnace walls by conduction and radiation. There are mechanical difficulties that interfere with the successful accomplishment of such a scheme, but the nearer it can be approached the greater will be the efficiency of the furnace and the process.

ELECTRODES.

Electrodes serve to carry current into furnaces in which electrical energy is converted in part to heat energy, and as the electrodes are the only avenue through which current is introduced into the charges, it is important that they should neither contaminate the charges nor cause undue waste of energy. They are made in various shapes and sizes to suit particular conditions, but the essential properties are common to all.

An electrode should possess high electrical and low thermal conductivity. It should be resistant to atmospheric oxidation and capable of withstanding handling and temperature changes without danger of breakage.

^a Stansfield, A., *The electric furnace*, 1907, p. 58.

Electrodes for use in electric furnaces are commonly divided into two classes, carbon and graphite electrodes. Metallic electrodes are employed in rare instances.

Carbon electrodes are made from such materials as petroleum coke (the ultimate residue in the distillation of petroleum), retort carbon (a pure form of carbon deposited in coal-gas retorts), and anthracite.

MANUFACTURE OF ELECTRODES.

The process of manufacture is simple. The material is highly heated in a retort to remove volatile matter, pulverized, mixed with a binder, formed in a press, and baked in a kiln. Graphite electrodes are simply amorphous-carbon electrodes that have been artificially graphitized in an electric furnace, and are thus a product of the electric furnace. Many of the difficulties and annoyances in the operation of large electric furnaces were until recently due to the breaking of poor electrodes. To-day strong electrodes may be had 24 inches in diameter, whereas only a few years ago none over 12 inches had the required strength.

CARBON COMPARED WITH GRAPHITE ELECTRODES.

There is no simple arbitrary rule by which to determine whether a carbon or a graphite electrode is the more suitable for any furnace. It is generally conceded that the graphite is more resistant to chemical action, and in particular to atmospheric oxidation. Moissan determined the temperature at which various forms of carbon begin to oxidize. According to him, amorphous carbon begins to oxidize at a temperature as low as 375° to 490° C., and graphite does not oxidize appreciably below 665° to 690° C. In practice, however, it is less apparent that graphite is inherently superior to carbon electrodes as regards oxidation. That the carbon electrode has the lower heat conductivity has been established by various workers in both experiment and practice. According to Wolgodine's^a research, the comparative conductivities of graphite and carbon expressed as temperature drop in centigrade degrees per kilowatt flow per square foot through a 1-inch thickness of material, are as follows: Graphite electrode, 2.3; carbon electrode, 4.7.

ENERGY LOSS IN ELECTRODES.

The total energy loss in electrodes is the sum of the loss due to the resistance offered to the passage of the current and the loss by conduction of heat away from the interior of the furnace. This amounts to about 8 per cent of the total input, and when cooling water is used this may account for another 3 per cent. The net efficiency of an

^a Wolgodine, S., and Queneau, A. L., Conductivity, porosity, and gas permeability of refractory materials: *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 383.

electrode, therefore, concerns both its electrical and its thermal conductivities. The electrode efficiency is consequently a physical characteristic for any electrode material, but varies in accordance with the relative importance of the electrical and thermal factors when the quality of the current impressed is varied.

EFFICIENCY OF ELECTRODES.

In general, the net efficiency is increased by producing the necessary wattage with as low a current and as high a voltage as possible. The difference in net efficiency of carbon and graphite electrodes need be only a few per cent. As graphite is a better conductor of electricity than carbon, the C^2R loss (C =capacity; R =resistance) is less in a graphite electrode than in a carbon electrode. The conductivity of graphite is usually taken as being four times that of carbon.

CONDUCTIVITY OF ELECTRODES.

The conductivity increases with increasing temperature. Misselt^a estimates that in general the thermal conductivity of insulating materials decreases at the rate of 1/273 part for each degree above 0° C. Small electrodes can be loaded more than large ones, possibly owing to the greater closeness with which the particles are packed. In view of this fact, modern factories have adopted increasingly larger presses in order to reduce this discrepancy and improve the larger sizes. Again, for the same composition, the specific resistance of round electrodes is much less than that of those with square cross sections. As to the ultimate sizes that can be produced, little can be said. As before stated, although the largest electrodes of sound quality formerly on the market were 12 inches in diameter, it is now possible to get them up to 24 inches without difficulty. Nevertheless, as has been pointed out by one writer on this subject, it may be doubted whether sizes will go on increasing, for the specific conductivity as well as the mechanical strength tends to decrease, whereas the difficulties in handling increase. Attempts have been made to increase the conductivity of electrodes by introducing metals, such as pure iron, with the carbon material while it is being molded. It is stated that the resistance thus formed can easily be reduced to one-half the ordinary value with comparatively little metal.

ESSENTIALS IN USE AND DESIGN OF FURNACE ELECTRODES.

As stated by Hering,^b the laws of electrode losses are:

1. The combined loss through the cold end of an electrode is equivalent to the sum of the loss by heat conduction alone (when

^a Misselt, Wichelto, Zeitschr. Ver. deut. Ing., June, 1908, p. 90.

^b Hering, C., Laws of electrode losses in the electric furnace: Electrochem. and Met. Ind., vol. 7, 1909, pp. 442, 514.

there is no current) and half the I^2R loss (I =rate of flow; R =resistance).

2. This combined loss will be least when the loss by heat conduction alone is made equal to half the I^2R loss; the total loss will then be equal to the I^2R loss, and no heat will be conducted from the interior of the furnace.

3. This minimum loss is dependent only on the material, current, and temperature, but not on the absolute dimensions; it merely fixes the relation of the cross section to the length, but leaves a choice of either; hence

4. For economy of electrode material the electrode should be made as short as practical considerations permit.

5. For each material there is a definite minimum loss of electrode voltage which depends only on the temperature and is independent of the dimensions or the normal current for which the furnace is designed; hence

6. The best possible electrode efficiency for any material may be determined from the total voltage of the furnace and this minimum voltage due to the material and the temperature, and is independent of the dimensions.

7. The temperatures indicated by the heat gradient of the combined flow are equal to the sums of those of the individual flows.

The proof of these laws is given in a paper by Hering on "Laws of Electrode Losses in Electric Furnaces"^a and need not be repeated here.

The fundamental principle involved is the one Hering announced in 1909, namely, that no heat should leave or enter the furnace through the electrode, or, in other words, that the heat generated by the electrical resistance shall raise the temperature of the hot end to that of the furnace; it is shown that under the given conditions this is also the condition of least total loss.

WATTAGE AS A MEASURE OF FLOW OF HEAT.

Before giving the formulas, Hering explained that it simplifies such calculations greatly to represent and measure a flow of heat in terms of the electric unit, the watt, instead of in calories per second. He stated that a watt is just as correct a measure of a flow of current of heat (calories per second) as an ampere is of the flow of electric current (coulombs per second). Heat is a form of energy, and the rate of flow of heat per second is power, hence it is measurable in units of power, such as watts. Since Hering called attention to this point others have endeavored to improve matters by calling this unit "watt-seconds per second," but this cumbersome name is entirely unnecessary and obscures rather than simplifies one's conceptions.

^a Hering, C., Trans. Am. Electrochem. Soc., vol. 16, 1909, p. 265.

It is evident that $\text{watts} \times \text{seconds} \div \text{seconds} = \text{watts}$. When the heat conductivity of a material is given as 10 watts for an inch cube, it simply means that with a 1°C . difference of temperature between two parallel sides, and perfect heat insulation on the other four, the same amount of energy will flow through as heat, say into water at the cold end, as would enter the water from a coil of resistance wire in which 10 watts were being set free.

For these reasons, in the following discussion, quoted from Hering ^a all flows of heat are represented in watts. Hering recommends that all thermal constants pertaining to electrodes be given in terms of watts instead of calories per second. The conversion factors are: Gram-calories per second $\times 4.18617 = \text{watts}$, and $\text{watts} \times 0.238882 = \text{gram-calories per second}$. The abstract from Hering follows:

FORMULAS.

The value of the symbols used in the following formulas are the same whether inches or centimeters are used, providing they are employed consistently throughout, including all those constants which are based on dimensions, and which of course will be different; those like the electrode voltage or watts per ampere are, of course, the same in both systems. All of the formulas in which the following symbols are used are in terms of actual units and may therefore be used directly in practice.

Let S =cross section in square inches.

L =length in inches (the essential length).

I =current in amperes.

W =watts generated electrically in the electrode.

H =heat flow in watts which would flow if there were no current.

h =heat flow in watts which enters the hot end from the furnace.

X =heat flow in watts leaving the cold end.

T =temperature drop in centigrade degrees between the hot and cold ends.

r =electrical resistivity in ohms, inch cube units.

k =thermal conductivity in watt, inch cube units.

e =electrode voltage in volts.

E =total voltage between the two ends, or the watts per ampere.

s =specific cross section in square inches.

S' =section in square inches per ampere per inch of length (or in square centimeters per ampere per centimeter length if the other quantities are in terms of centimeters).

DETERMINATION OF PROPER SIZE OF ELECTRODES.

The underlying feature is that the electrode shall not chill the furnace nor develop a high temperature point within the walls at which the temperature is greater than in the furnace. Hence the hot-end temperature should be as nearly as practicable equal to that of the furnace.

The current and furnace temperature are always given in the specifications. The material and either the length or the section (but not both) may also be specified, but preferably all three should be left to the designer. The voltage and loss can not be specified, as they are determined by the current, temperature, and material.

The length (that is, the essential length, not including additions to either or both ends for other purposes than to get the energy through the walls) may be determined solely by the thickness of the furnace walls; but as this affects the cross section, some

^a Hering, C., loc. cit.

latitude should be left in case the corresponding section is found to be too large or too small to be practicable.

The section in square inches is then determined at once by multiplying the proper temperature value of S' from the table [“constants for calculating electrodes”] by the current in amperes and by the length in inches. Formula: $S=ILS'$. If this is too large or too small to be practicable, the length may be changed and the section may be redetermined. The quotient of the section divided by the length is a constant for any specified conditions (numerically equal to S' from the table multiplied by the current), hence they may both be increased or decreased in the same proportion.

Should this table of values not be available and the specific section be known, formula $S=ILs\sqrt{T}$ should be used for calculating the section. If this is also not known, then use the conductivities in the formula $S=IL\sqrt{r\div2kT}$. Great accuracy is not necessary when one is near the correct result, as an error in section near that point produces a relatively much smaller error in the loss.

The loss in the electrode in watts is entirely independent of the section or length adopted, provided only that their quotient is approximately as above. It is calculated in watts by multiplying the corresponding temperature value of E in the table by the current (formula $mX=IE$). If this table is not available, we may use the electrode voltage in the last part of formula $mX=2H=W=I\sqrt{2krT}-Ie\sqrt{T}$. If this is not available either, then use the conductivities in the preceding expression of this formula. This loss is for one electrode and it is the least possible under those conditions.

Constants for calculating electrodes.

Temperature.		E. (Watts per ampere.)				S'. (Square inches per ampere per inch length.)				S'. (Square centimeters per ampere per centimeter length.)			
°C.	°F.	Car- bon.	Gra- phite.	Iron.	Cop- per.	Car- bon.	Gra- phite.	Iron.	Cop- per.	Car- bon.	Gra- phite.	Iron.	Copper.
400	752	1.00	0.85	0.145	0.095	0.00165	0.00040	0.000103	0.000016	0.0042	0.00100	0.00026	0.000041
600	1,112	1.38	1.04	.225	.140	.00114	.00031	.000102	.000015	.0029	.00078	.00026	.000037
800	1,472	1.68	1.19	.295	.184	.00090	.00027	.000101	.000014	.0023	.00068	.00025	.000036
1,000	1,832	1.93	1.32	.350	.226	.00076	.00024	.000098	.000014	.0020	.00061	.00025	.000036
1,200	2,192	2.15	1.43	.410	.270	.00069	.00022	.000094	.000014	.0018	.00056	.00024	.000035
1,400	2,552	2.36	1.52	.460	.310	.00062	.00020	.000092	.000014	.0016	.00051	.00023	.000035
1,600	2,912	2.53	1.62	.510	.351	.00057	.00019	.000089	.000014	.0014	.00048	.00023	.000035
1,800	3,272	2.68	1.71	.565	.391	.00053	.00018	.000087	.000014	.0013	.00046	.00022	.000035
2,000	3,632	2.83	1.79	.610	.430	.00049	.00017	.000085	.000014	.0012	.00044	.00021	.000035

When the current for a furnace varies appreciably, the calculations must of course be made for some assumed normal value, remembering that whenever it is less than that the electrode will chill the furnace more or less; and whenever it is greater the electrode will get hotter within the walls and in both cases the total loss will be greater.

In operating a furnace, if the electrode is found to chill the product it is either too short or too large in section. If, on the other hand, it is found to produce excessive temperatures within the walls, it is too long or too small in section. The current density is not a determining factor.

The hotter the outside terminal is allowed to get, the smaller the loss, but the larger the section or shorter the electrode.

When the proportions turn out to be large in section and short in length the relative proportions may be improved by increasing both length and section. On the other hand, if the electrode is abnormally long and small in section (as for instance with iron or copper and small currents), then if the section can not be made smaller, there seems to be nothing left to do except to sacrifice some of the loss by making the electrode shorter and it will then chill the furnace more or less.

The additional lengths of the electrode necessary for the terminals, for feeding, or for the distribution of the current in the inside, must be calculated separately; as they are determined by entirely different laws and conditions; the above refers only to the essential part which is necessary to get the current into and out of the furnace as well as possible. The above length must, therefore, never include the long external part outside the furnace for feeding purposes. Such a part radiates heat to the air, and therefore follows entirely different laws of proportions. A redeeming feature of such a case is that the outside temperature used to determine the proportions of the essential part, may be allowed to be very high, probably limited chiefly by oxidation, thereby reducing the total loss which has been increased by the long external part.

PROPER CONSTRUCTION OF ELECTRODE HOLDERS.

HEAT SOCKETS AND CLAMPS.

When the current is fed into the end of the electrodes, as in some carbide and ferro-alloy furnaces, the connections are applied by means of heat sockets whereby they are frequently attached once for all. The other method is to use clamps which permit the electrode to be passed through them, their clutches being merely pressed tightly against the smooth hard furnace electrode.

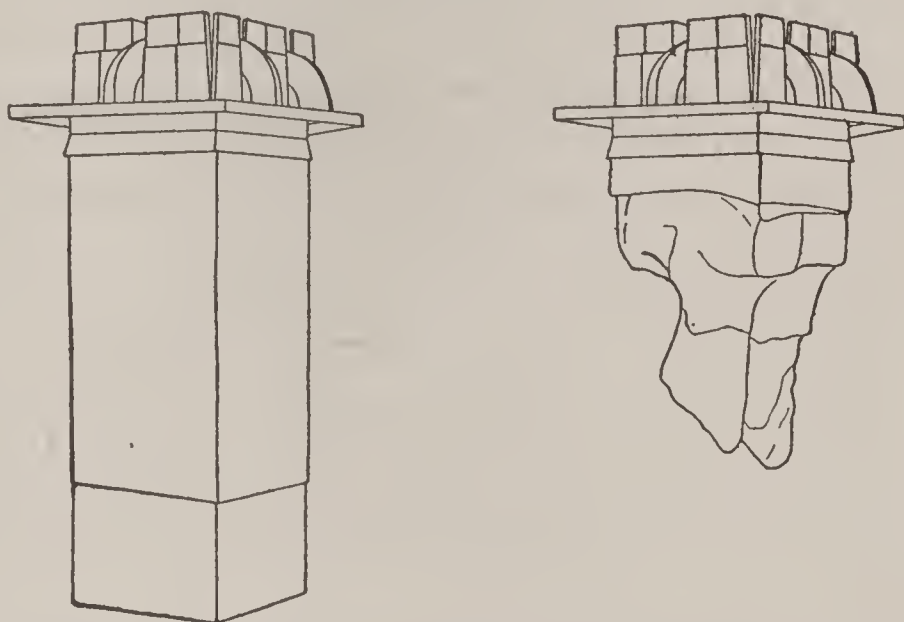


FIGURE 14.—Trollhättan electrode before and after use. Old type of holder.

The latter is preferred as a means of conducting the current to the electrode, owing to the fact that thus it is possible to place the "electrode holders" immediately on the roof of the furnace and keep them stationary. This arrangement permits the introduction of the current as close as possible to the end of the electrode where it enters the charge, so that less energy is spent in overcoming the resistance of the electrodes themselves than when the current is introduced at the end of the electrode. The connection between the cables and bus bars carrying the current to the furnace, and the electrodes, which, as before stated, serve to carry the current into the furnace, is made by electrode holders. These holders for convenience may be divided into two classes, namely, top holders and side holders.

TOP HOLDERS.

With top holders the current is introduced at the top of the electrode through a holder such as is shown in figure 14. The holder attached to the end of the electrodes is also shown in figure 15; and,

as will be noted by reference to the figure, the same contact surface is maintained between the holder and the electrode until the electrode is used up. Top holders are generally used in open-top furnaces, such as the ferrosilicon and carbide furnaces, and in all furnaces in which a frequent raising and lowering of the electrode is necessary. If the electrode as a whole is subjected to erosion from any cause, it is then practically necessary to use a top holder, owing to the fact that the uneven surface produced by the erosion would prevent a quick and effective fitting of the electrode holder to the electrode, especially after the electrode became hot, and the proper contact between the holder and the electrode is, as can be readily understood, of great importance.

SIDE HOLDERS.

Side holders are generally used on closed-top furnaces or on constructions in which the electrode above the holder will not be subjected to sufficient heat to bring about its oxidation and thus cause it to have an irregular surface. Electrodes used with side holders must be of fairly uniform dimensions. For this reason, graphite electrodes are better adapted for use for side holders than are amorphous-carbon electrodes, as they are more easily machined. This is especially true where it is desired to keep the top of the furnace not only closed but tight, as for example in the Swedish and California types of iron reduction furnaces. In these types of furnaces the electrode holder is placed on the roof of the crucible, for in this way the current is conducted into the electrode at a point where it will have the shortest distance to travel before it reaches the point where it does its work, and in this way the electric-energy loss that may be due to the resistance of the electrode is reduced. Then, too, it is to be remembered that the ohmic resistance of carbon decreases with increasing temperature, and as the hottest part of the electrode will be that part that extends down into the furnace, the ohmic resistance of the electrode is reduced to a minimum by getting the holder as near the hot end of the electrode as possible. The side holder permits of the joining of one electrode to the other without the removal of the holder, an arrangement that is, of course, decidedly advantageous.

ESSENTIAL DETAILS OF HOLDERS.

In the construction of electrode holders all current-carrying parts are usually of copper or bronze, the main posts and supports being of steel. A current density of from 19 to 30 amperes per square inch is usually adopted for the current-carrying contact parts, the lower limit being preferred.

As before stated, the surface condition of the electrode is of the greatest importance, so far as the making of a good contact between

the holder and the electrode is concerned. If the contact is not a good one the holder becomes hot and may even be ruined. On small furnaces not equipped with water-cooled holders it is especially necessary to have a good contact between the holder and the electrode, and so elastic layers of copper netting, thin iron sheets, or something of the kind, is used to insure a good contact. Efficiency, of course, demands a good contact between the electrode and the holder, even if the holder be water-cooled, for although the water cooling may prevent the holder from becoming excessively hot, the heat developed is at the expense of electrical energy, and is simply carried away by the cooling water without having done useful work. Then, too, a bad contact is likely to cause the electrode itself to become red-hot adjacent to the holder, and thus cause a serious deterioration of the electrode, due to burning; and the excessive heat from the electrode may cause a water-cooled electrode holder to crack or break, and because, as before stated, ohmic resistance decreases with increasing temperature, the flow of current will be greatest through the hot part of the electrode and will cause this hot part to become still hotter.

As illustrating the importance of using the best type of electrode holder possible, the following figures are taken from an article that appeared in "Stahl und Eisen,"^a a translation of which has been published in Metallurgical and Chemical Engineering.

In a large plant producing 4,000 tons of ferro-alloys per year the cost of maintenance of the holders was marks 6.40 (\$1.60) per ton of metal produced, or marks 25,600 (\$6,400) per year. By using an improved holder this cost was reduced to marks 1.60 (\$0.40) per ton, saving marks 19,200 (\$4,800) annually. The consumption of electrodes was also reduced, resulting in an additional saving of marks 12,800 (\$3,200). The total saving due to the use of an improved holder was, therefore, marks 32,000 (\$8,000), or marks 8 (\$2) per ton of product.

SPECIALLY SHAPED ELECTRODES.

Owing to the fact that it is important to effect the proper contact between the electrode holder and the electrode, it was rather common practice to shape the ends of the electrodes to fit the holders, especially where top holders were used, and as good amorphous carbon is generally difficult to machine properly, the particular shape desired was given to the electrode before baking. Some of these special shapes are shown in figure 15.

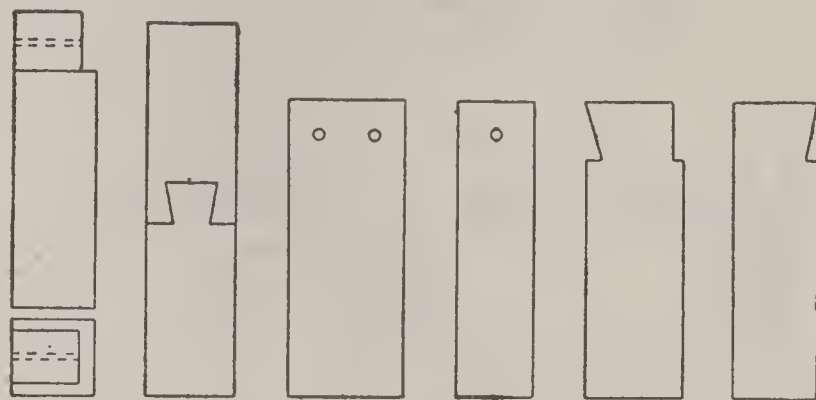


FIGURE 15.—Specially shaped electrodes for electric furnaces.

^a Anon., Die Electrodenfassungen bei Electroöfen: Stahl und Eisen, vol. 33, 1913, pp. 472-478, 555-561. Translated by Kennelly, A. E., Electrode-holder construction for electric furnaces: Met. and Chem. Eng., vol. 11, June, 1913, p. 321.

TYPES OF TOP HOLDERS.

The subject of electrode holders has been discussed in "Stahl und Eisen,"^a and a translation of the article has been published in Metallurgical and Chemical Engineering.^b The following lengthy quotation from this translation is given here:

In carbide plants the connection shown in figure 16 was formerly often used, consisting of two solid V-shaped cast-iron clamps *a*, tightened by bolts *b*, and connected to the cables by means of necks *c* and pressure plates *d*. This solid electrode holder never proved satisfactory. When the electrode became short the holder as well as the electrode became hot and the cast-iron pieces often melted. Later on, a cooling device was combined with this construction, as shown in figures 17 and 18.

Figure 19 shows a connection that has proved satisfactory in a carbide furnace. The head of the electrode is dovetailed and clamped between two cast-iron plates *b* by means of two weights *a*. The whole apparatus is supported by an iron structure which itself hangs on an electrically insulated hook. Copper plates *d*, *e*, and *f* serve to make contact at *h* with the flexible laminated ribbons *g*, carrying the current. The mechanical suspension is independent of the electric connection, and the weight of the electrode tends to tighten the contact between copper plates and electrode head. An electrode equipped with this holder can be put into service very quickly by hooking into the chain of the windlass and making contact at *h*.

The holder shown in figure 20 has been successful in the manufacture of ferro-alloys. The electrode consists of four carbon blocks embedded in a tamped carbon mixture. Two opposite surfaces of the same block have dovetailed recesses in order to clamp the copper plates *a* by means of the screw bolts *b* and the plates *c*. This holder is simply connected to the plates *d* of the current lead *e*. The

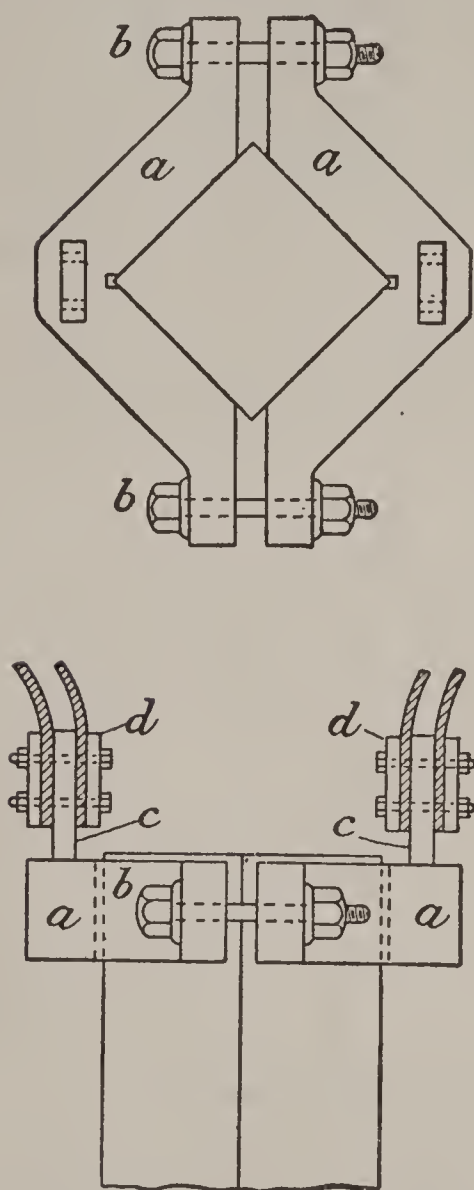


FIGURE 16.—Holder formerly used with carbide furnaces.

plates *a* preferably consist of copper, the plates *c* of steel casting, the pressure screws of iron with square threads, and the plates *d* of bronze or of cast iron containing manganese.

If the surface of the dovetails is not smooth enough to secure a good contact, the heating at the contact will be avoided by putting a cushion of very fine strands of a copper cable between the carbon and the copper plate. Under the pressure of the screws this cushion of about 0.5 cm. ($\frac{3}{16}$ inch) thickness is pressed into the rough spots of the carbon and secures the passage of the electric current under favorable conditions.

When the electrode butts are to be utilized down to a length of 50 cm. (20 inches) it becomes necessary to cool the parts artificially to prevent the holder from being destroyed by the heat of the furnace. Figure 21 shows the copper plates of the current lead being wedged to the carbon by means of water-cooled channels, *a*, open at the top. This design has given good results, especially in furnaces with an open hearth.

^a Anon., Die Electrodenfassungen bei Electroöfen: Stahl und Eisen, vol. 33, 1913, pp. 472-478, 555-561.

^b Issue of June, 1913, pp. 321-324.

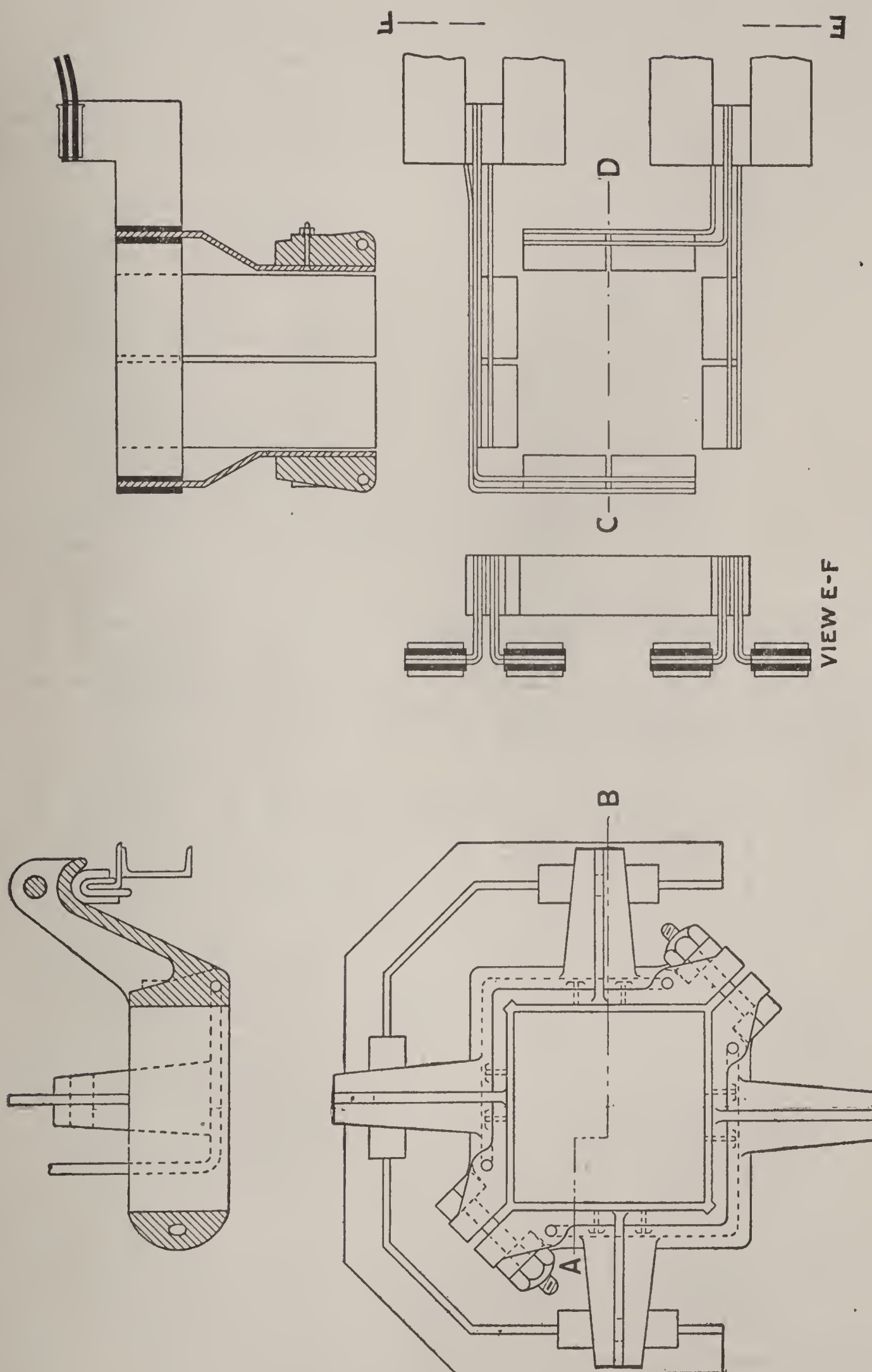


FIGURE 17.—Electrode holder with cooling device.

The holder shown in figure 22 is good for cylindrical electrodes. The flexible leads are fastened to a bronze piece, *a*, of special form. This piece is fastened to the carbon by an electrically insulated ring and has a contact surface machined on the lathe. The holder itself is a box, *c*, surrounding the head of the electrode and pressed to the piece *a* by the screws *b*. After loosening the screws *b*, the pieces *a* and *c* can be taken apart by a quarter turn. The piece *c*, which is protected by the iron sheet *d*, is put on top of the electrode, and bronze, *e*, is poured into the free space between the box and the carbon and on top of the electrode.

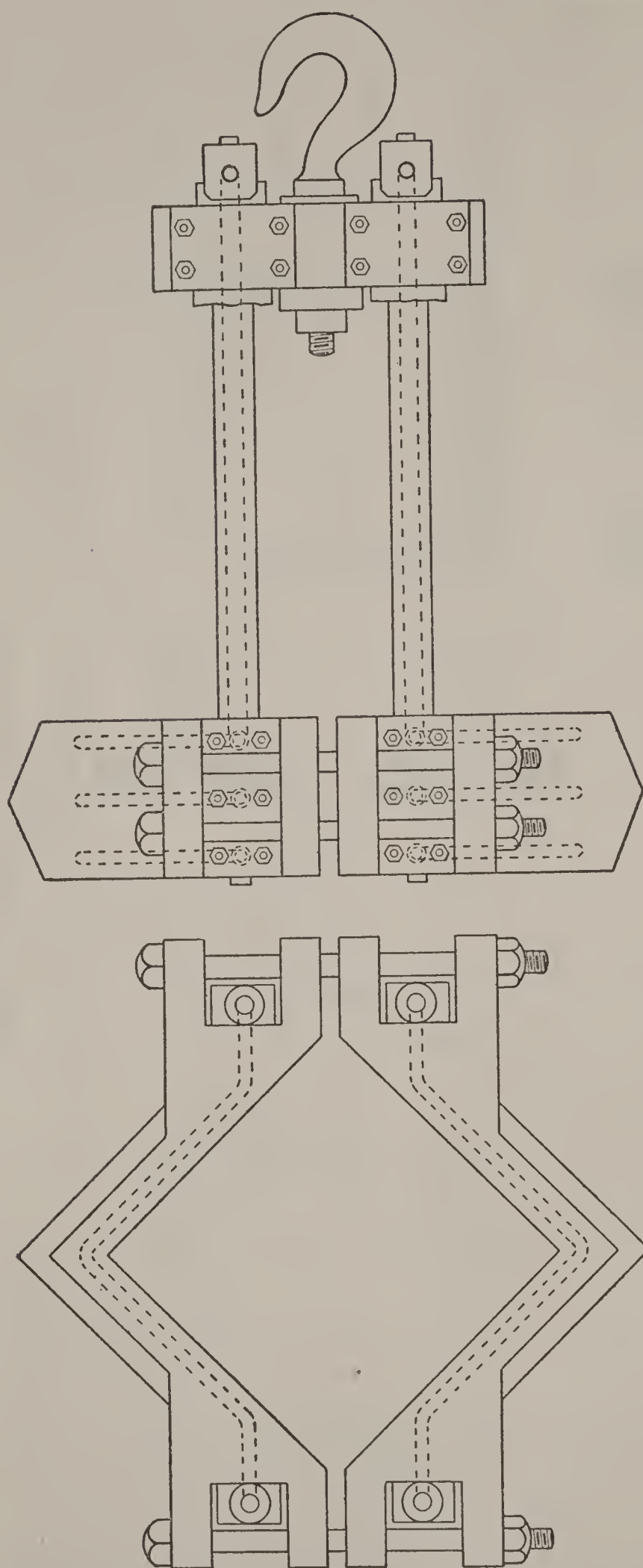


FIGURE 18.—Electrode holder with another type of cooling device.

of the electrode can be ground into each other, which warrants a good contact on the entire surface. As it is hardly ever possible to get the surface of a nonmetallic electrode

by an electrically insulated ring and has a contact surface machined on the lathe. The holder itself is a box, *c*, surrounding the head of the electrode and pressed to the piece *a* by the screws *b*. After loosening the screws *b*, the pieces *a* and *c* can be taken apart by a quarter turn. The piece *c*, which is protected by the iron sheet *d*, is put on top of the electrode, and bronze, *e*, is poured into the free space between the box and the carbon and on top of the electrode.

The weight of the electrode produces an excellent contact between the bronze and the walls of *c*. The holder is cooled with water which enters through *f*, and drops through the central overflow pipe onto the top of the electrode; the admission of water is regulated to make up for evaporation.

Figure 23 shows a holder cooled by water under pressure. The ring-shaped piece *a* is hollow in the center, the water circulating through *b*. Copper sheets, *c*, serving both as mechanical suspension of the electrode and as current leads, are pressed against the electrodes with cast-iron or bronze wedges, *d*. The wedges have regulating screws, and piece *a* is held by four pressure screws.

While an intimate contact is hard to make between a carbon electrode and the cable, it is easily obtainable between the cable and a metallic electrode by screwing the pole shoe at the end of the cable to the electrode. The contact surfaces of the pole shoe and

perfectly plane there will always be a certain contact resistance and evolution of heat and it may be necessary to remove this obnoxious heat of the contacts by special contact-cooling devices.

The Westdeutsche Thomas Phosphatwerke have a German patent (No. 207,361) for the construction shown in figure 24. The square-shaped electrodes *a* have a recess on top, and a copper cap *c* is dumped over this head. In order to make good contact where the surfaces do not touch each other, the cap is filled with liquid aluminum or another metal of high melting point, and while the aluminum (or other metal) is still liquid a

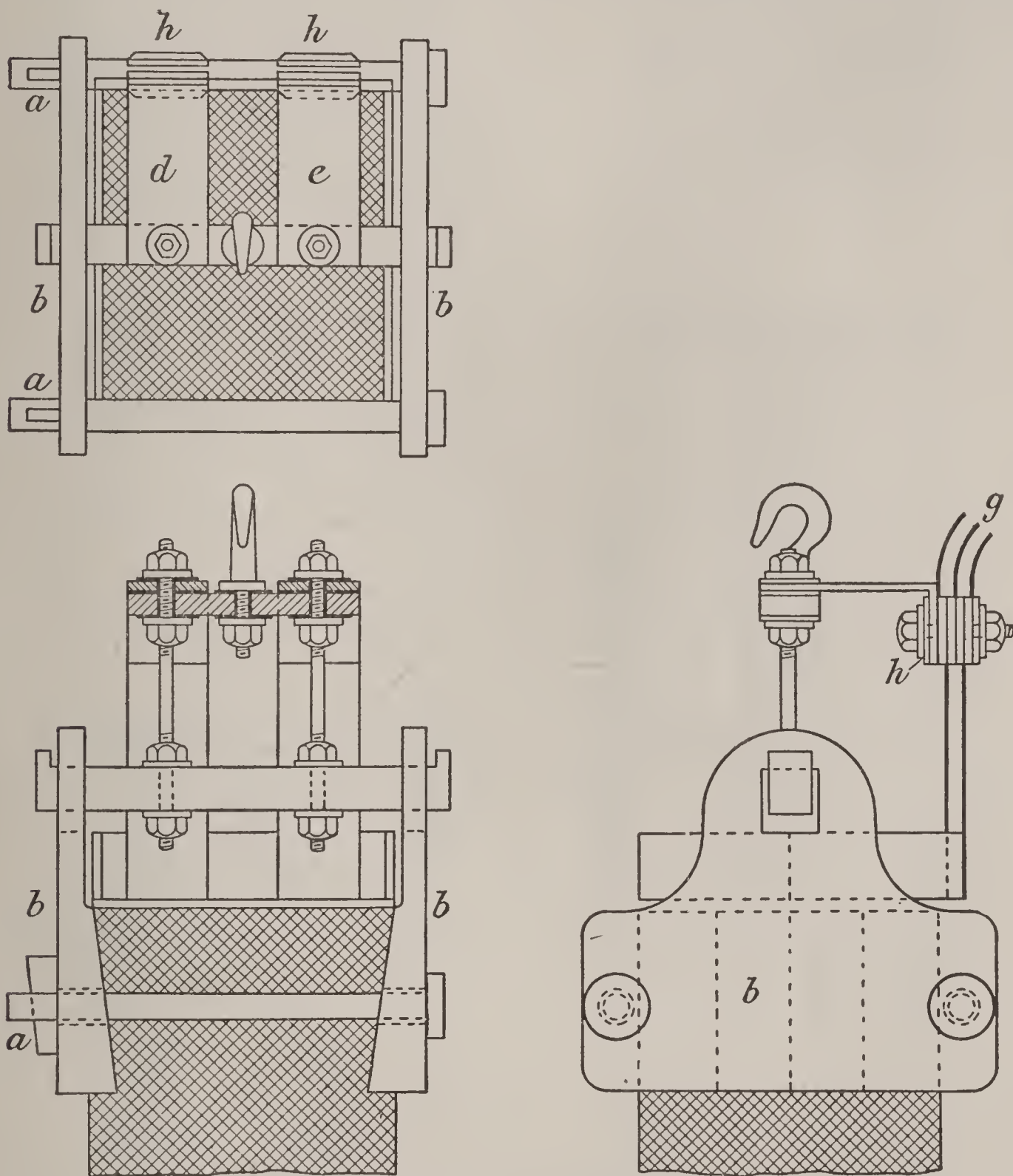


FIGURE 19.—Detailed views of electrode holder satisfactorily used in carbide furnaces.

red-hot wrought-iron ring *e* is shrunk around the copper cap. By the simultaneous gradual cooling of the aluminum and the iron ring a very close contact is produced between the copper cap, the aluminum layer, and the top of the electrode.

Contact resistances can not show up any more. The pole shoe *f*, the screws *g*, the hoops *l*, and the cable ends can be seen from the figure. As no soldering is required in this design, an important source of trouble is eliminated.

With the patterns of holders described so far, the remaining butt of the electrode, which is about 50 cm. (20 inches) long in furnaces with an open hearth, is not used.

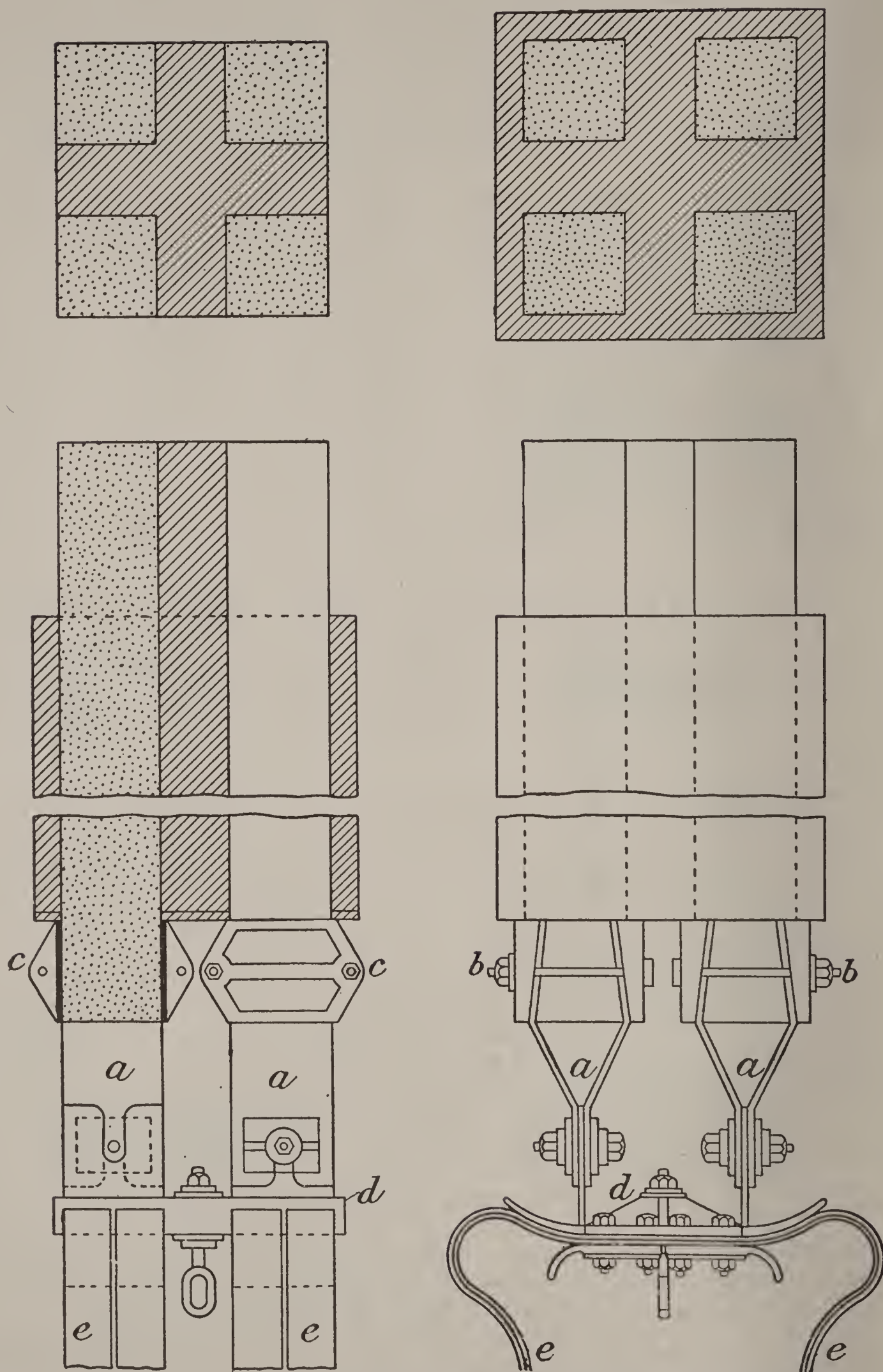


FIGURE 20.—Another type of electrode holder satisfactorily used in carbide furnaces.

In furnaces with arches, as for instance, in electric steel furnaces, the discard is larger, leaving a butt of about 90 cm. (36 inches), which means that only about one-half of the electrode can be used.

MOVABLE SIDE HOLDERS.

In order to utilize completely the electrode material, movable holders are employed which allow the electrodes to slide through. The electrode is not held on one end, but

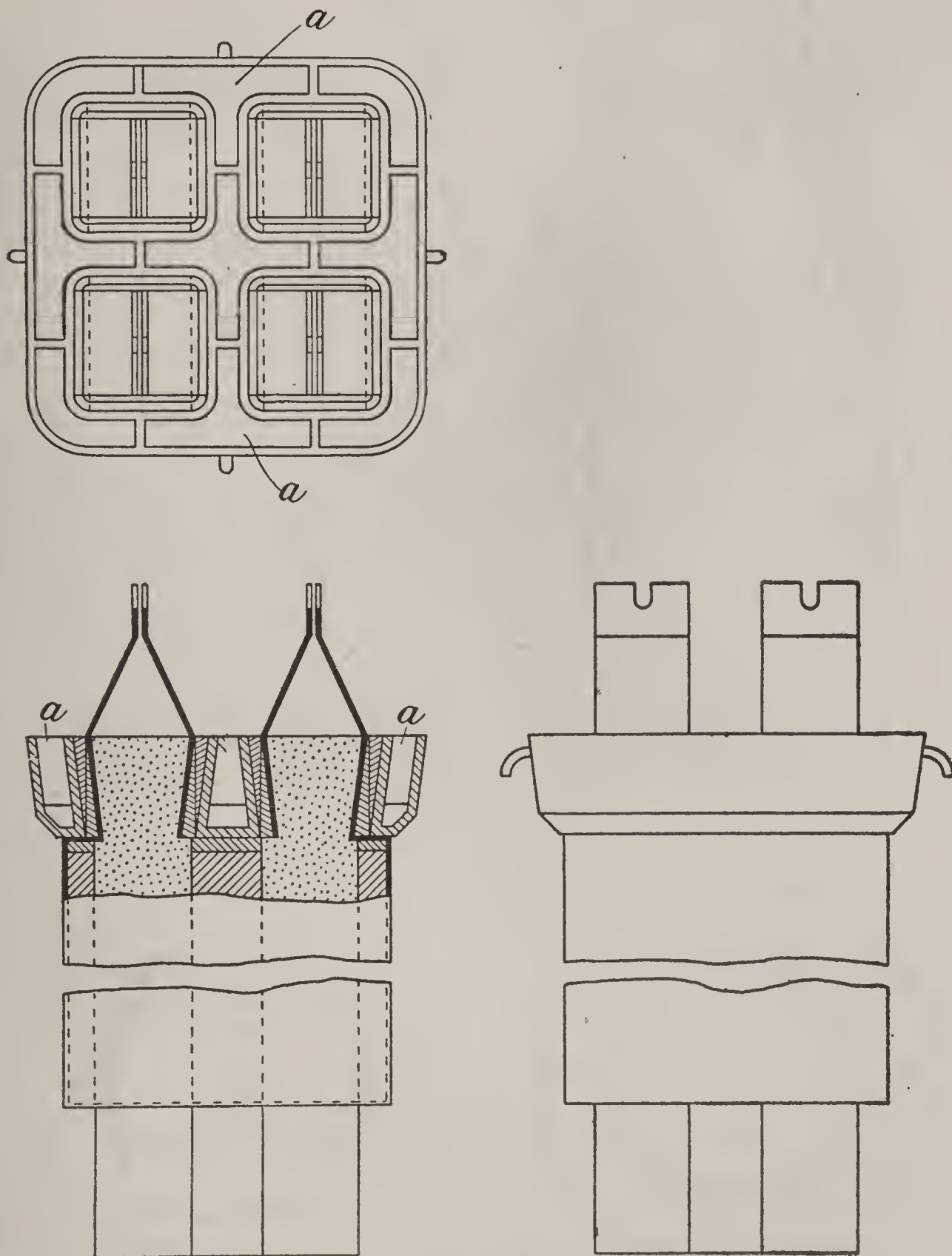


FIGURE 21.—Water-cooled electrode holder.

somewhere between its two ends connection is made to the cable; when a piece of electrode is used up, another piece of carbon is added to the previous piece, and the holder raised accordingly.

Figure 25 represents this type of connection in a design proposed by Louis. The electrode is held between two hollow pieces of metal, *a*, which form a pair of tongs.

The tongs are pressed against the electrodes by the screw *b*. When the lower part of the electrode is consumed, it can be fed downward for the desired length. The cable is fastened at *c*; the best way of cooling the pieces *a* and the contact is by means of water under pressure.

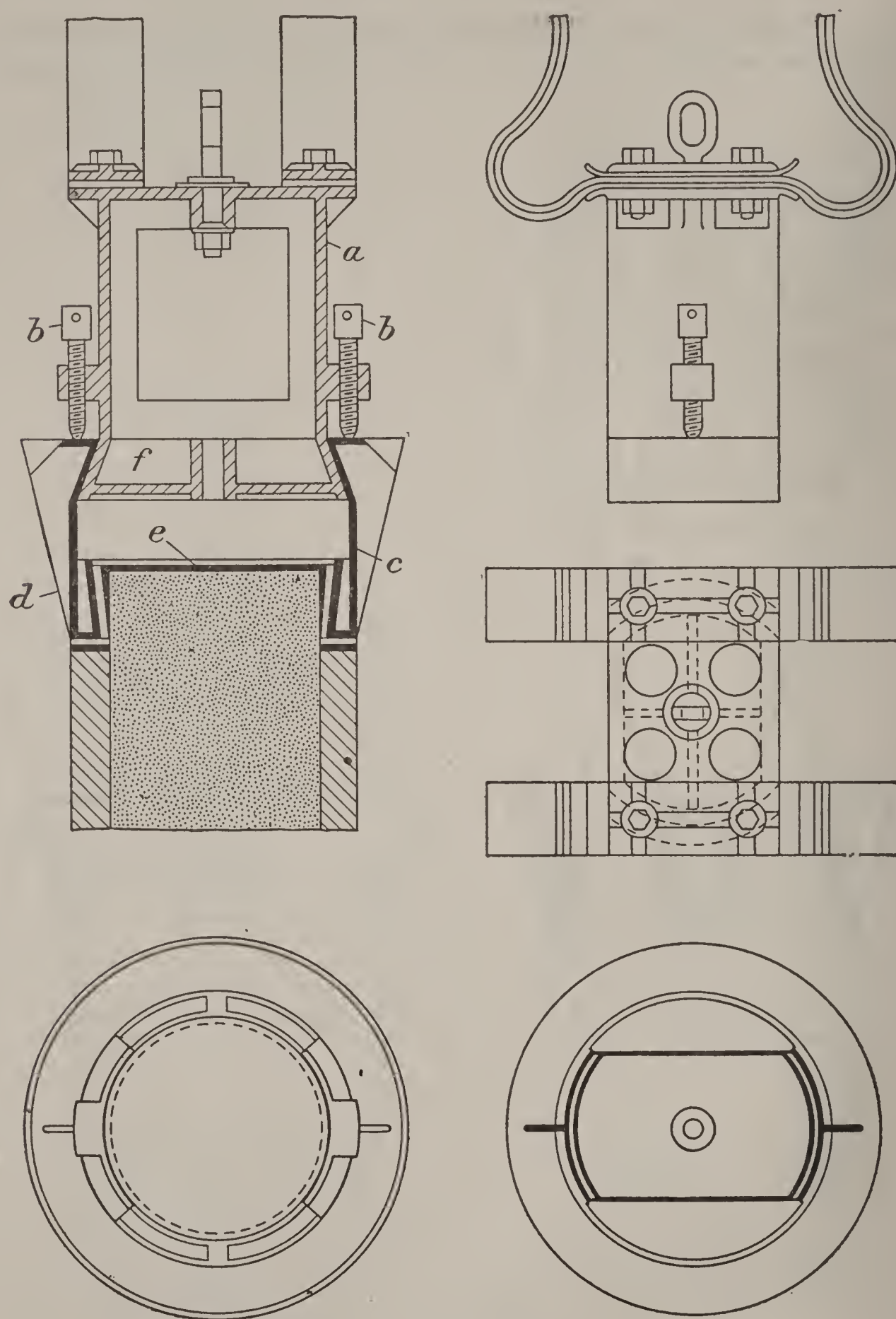


FIGURE 22.—Holder for round electrode.

Figure 26 shows another tongs design for a combined electrode of several blocks. The carbon blocks are pressed by steel levers by means of the screws *a* against the central piece of bronze, which serves as suspension and current lead.

Figure 27 shows a similar design in which adjustable wedges are employed which are water cooled.

The holders of most electric steel furnaces are built for complete consumption of the electrode material and have been repeatedly shown to our readers.

In the Héroult furnace each electrode is held by a bracket from a vertical rack, which is moved up and down by gear and pinion. Each carbon is surrounded by a band of metal, pieces of copper being inserted, which carry the current from the cables.

Figures 28 and 29 show various constructions used by the Aktisbolaget Elektrometall Ludwika, and by Nathusius.

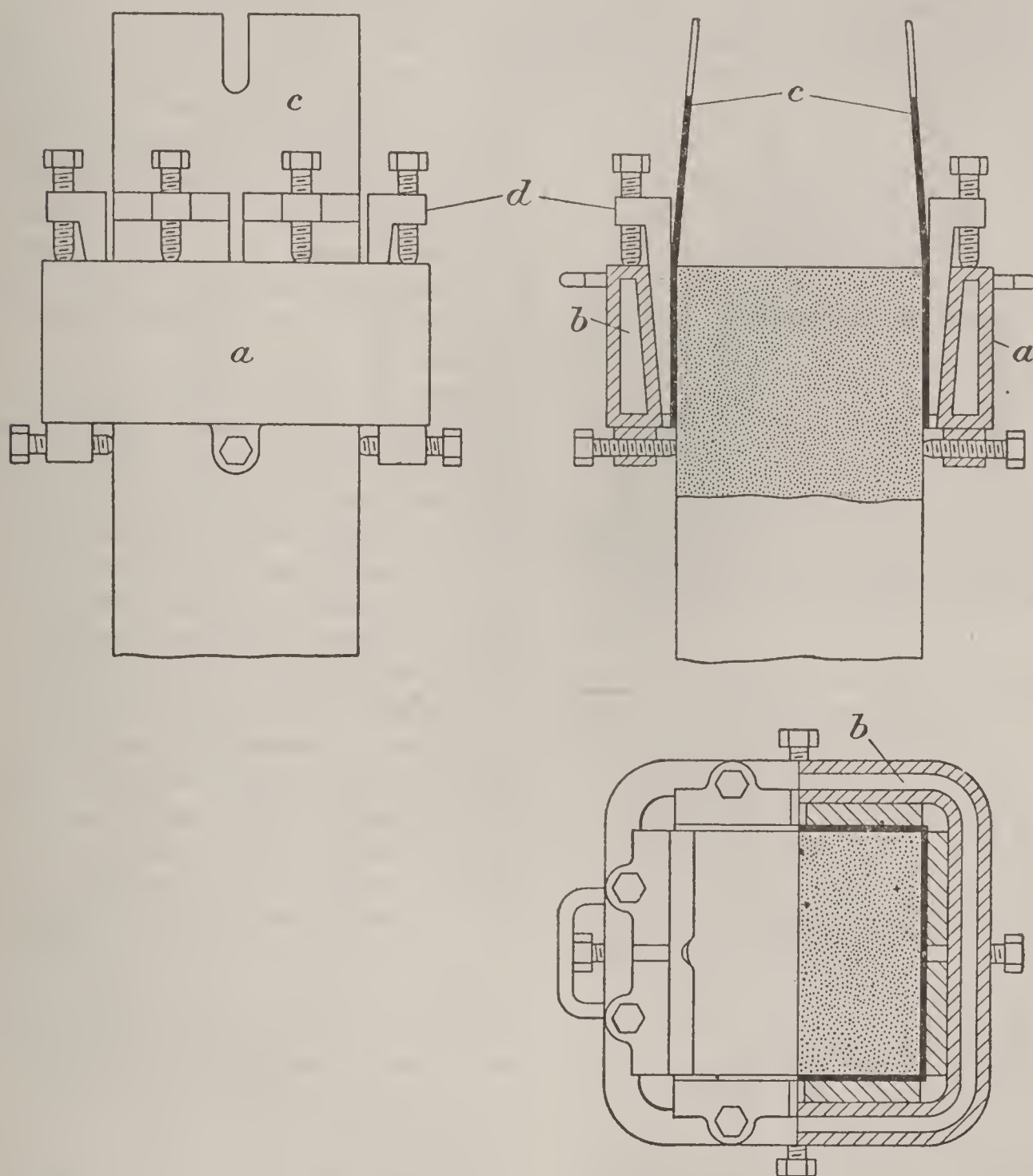


FIGURE 23.—Electrode holder cooled with water under pressure.

METALLIC CONDUCTORS IN ELECTRODE HEADS.

Keller's method of inserting the metallic conductor into the head of the electrode gives satisfactory results. This method has been known for some time. With small electrodes the connection to the current leads is obtained by means of metallic plates between the conductor and the carbon. But too close a tightening of the bolts often causes the electrode to crack whereas otherwise the contact may be poor.

Figure 30 shows an electrode connection often used for the manufacture of aluminum. The end of a flat piece of iron is upset in the shape of a fishtail and then molded into the

electrode. Plants who manufacture their own electrodes insert this connector before baking, whereas those who buy them baked fasten the metal with a graphite mortar or by pouring liquid bronze into the hole.

Figure 31 shows a connection made by a screw bolt. A thread is made in the end of the electrode, and an iron rod, threaded at one end, is fastened in the same. A mantle of bronze or aluminum bronze forms contact at both ends and secures the passage of current. Four or six of these electrodes are sometimes combined in aluminum furnaces, as shown in figure 32.

A connection of Dr. Lessing, of Nuremberg (German patent 211,273), is shown in figure 33. It consists of a piece of iron molded with the electrode which has square-shaped threads of such an angle that the distance b between the threads is considerably

larger than their thickness a . This bolt can be removed from the carbon before baking without destroying the threads in the latter. The iron piece d connects with the threaded bolt c . By screwing the bolt into the carbon and securing it by a counternut, e , an excellent contact is obtained.

Keller allows the metallic conductor a certain play, the lower end being tapered thicker into the hollow top of the carbon (German patent 218054; fig. 34). Liquid metal, copper or pig iron, is then poured into the space left in between. This produces a mechanically and electrically reliable contact. No jaws, wedges, or screws are required to make or improve the contact. The interior of the metallic conductor is intensely water-cooled in order to utilize as much length of the electrode as possible. The metal has to be poured carefully into the open gap between conductor and carbon. The cavity in the electrode is first heated by a piece of red-hot iron, the preheated current conductor is then introduced and kept about 5 mm. (0.2 inch) distant from the bottom of the cavity by a small, hard body. Very hot bronze or cast iron is then poured in.

Although the metal shrinks a little, it makes an intimate contact. The effect of

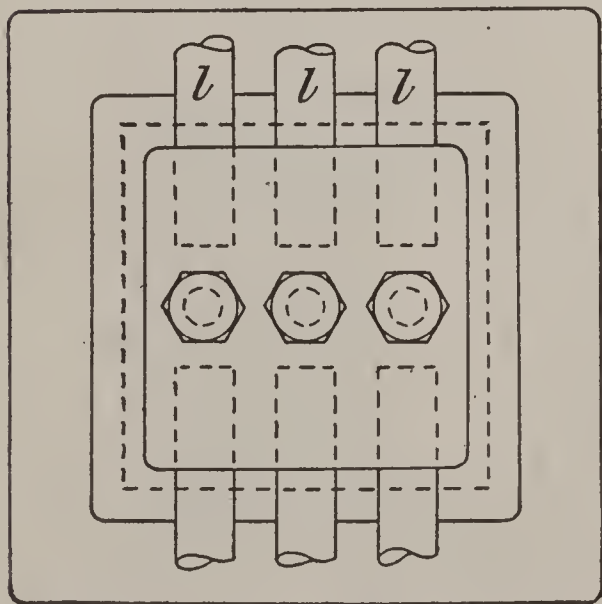
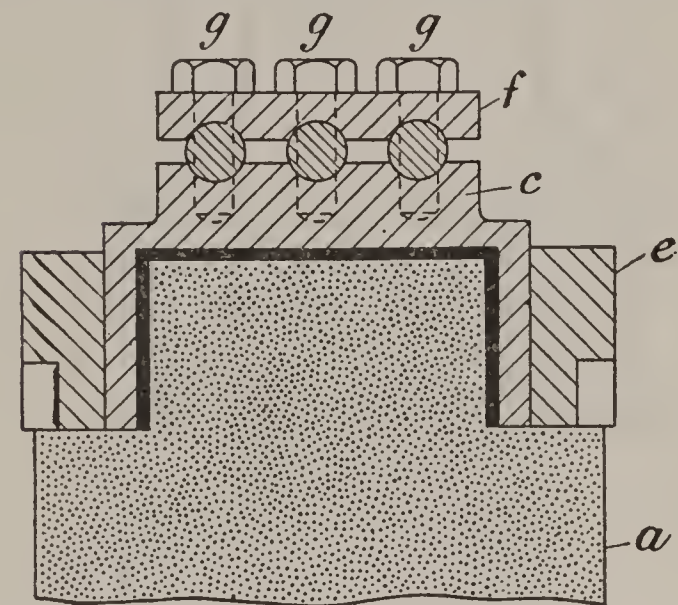


FIGURE 24.—Electrode holder used in Germany.

shrinkage can be entirely overcome by casting about 30 grams of tin around the contact metal after it has cooled. If the contact should not be perfect, it heats up when the electrode is put to use, the tin melts and flows into the interstices and makes a perfect contact. The costs of this method are said to be low. The contact metal and the tin can be recovered after the electrode has been consumed.

This type of connection is mechanically strong, and its electrical resistance is low. As the cooling effect reaches clear down into the bottom of the cavity in the electrode, the contact will not soften, even when it is heated by the proximity of the arc. The electrode can, therefore, be introduced through the roof of the furnace and be used up to an extremely small rest piece. As no part of the metallic conductor is wider than

the electrode, it can pass through the same opening which fits closely around the carbon and can be packed in firebrick up to the diameter of the electrode. This method is satisfactory for single electrodes, as well as for bundles of them.

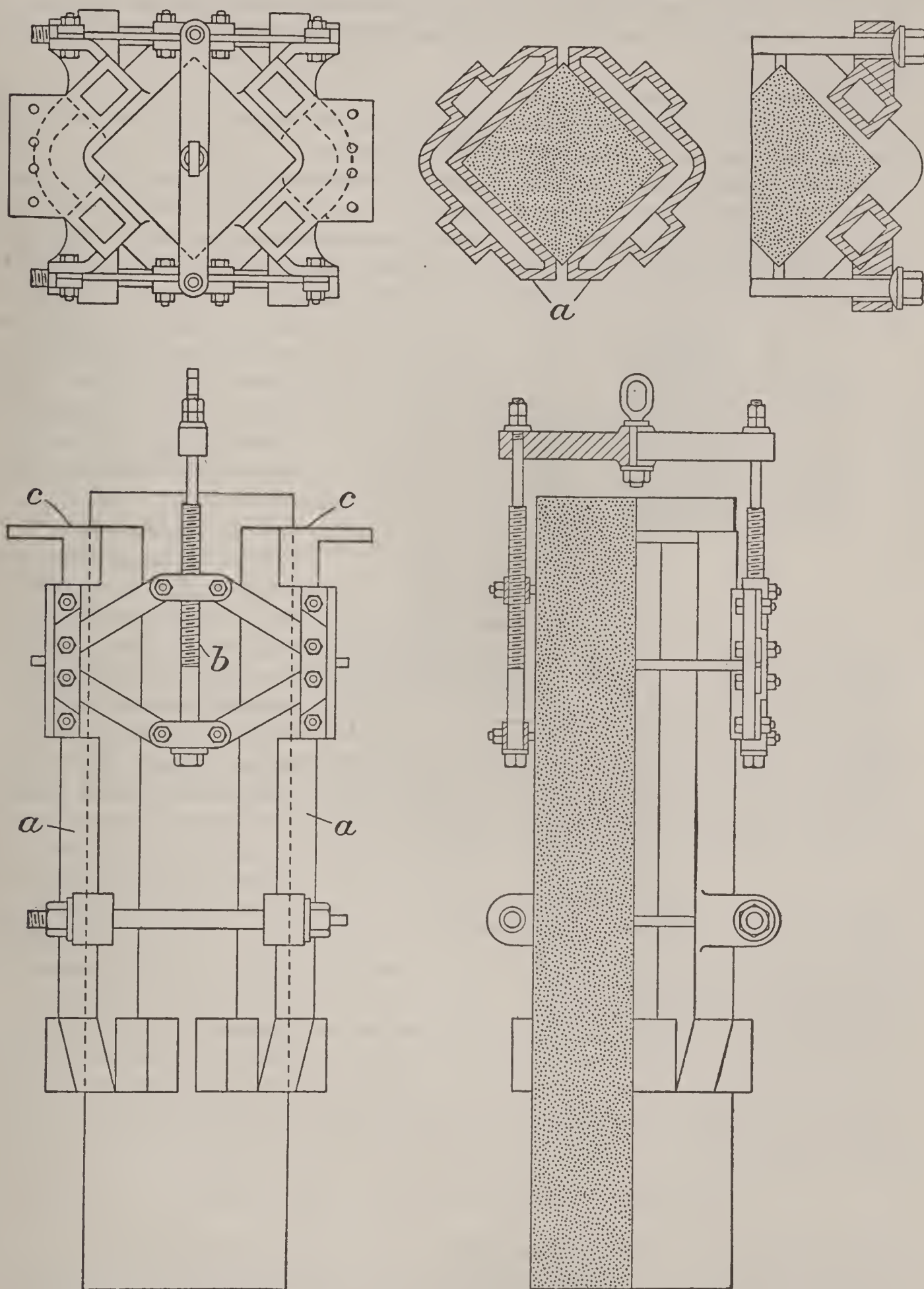


FIGURE 25.—Electrode holder with movable contact.

The advantages of Keller's connection are:

1. It avoids all intermediary contacts between the current cable and the carbon.
2. It avoids all parts extending beyond the area of the electrode, and has no brackets, screw bolts, wedges, etc., which are liable to deteriorate and cause a high maintenance cost.

3. It requires no finishing of the electrode surfaces or of the metallic conductor. There is even a certain advantage in leaving the walls of the cavity roughly finished, as it improves the contact surfaces and the mechanical strength of the joint.

4. The electrical contact is perfect, as the liquid metal fills all the free spaces.

5. The mechanical connection is strong and fool-proof, even when the carbon is almost used up.

6. The cost of maintenance is low.

7. The electrode can be used down to below 25 mm. (less than 1 inch) length, which dispenses with the cumbersome joining of electrodes.

A practical elastic connection between the electrode and the current leads is used by Keller with the above-mentioned electrodes (German patent 194897). The successive up-and-down movement of the electrodes requires flexible leads between the conductor and the carbons. Sometimes an elastic cable of sufficient length serves this purpose. This, however, is subject to considerable wear and tear, takes much

space, and is cumbersome for melting furnaces with vertical electrodes, especially so when they lead the current in and out, so that there is the danger of short circuit.

Keller avoids these difficulties by using thin and very flexible leaf springs of copper, for example, 0.5 mm. (20 mils) thick, bundled into two symmetrically arranged bundles and held together at several points by rings. Each side of the electric holder has a guiding rod, which causes the leaf springs to be compressed or opened in vertical direction when the electrodes move down and up. The shape

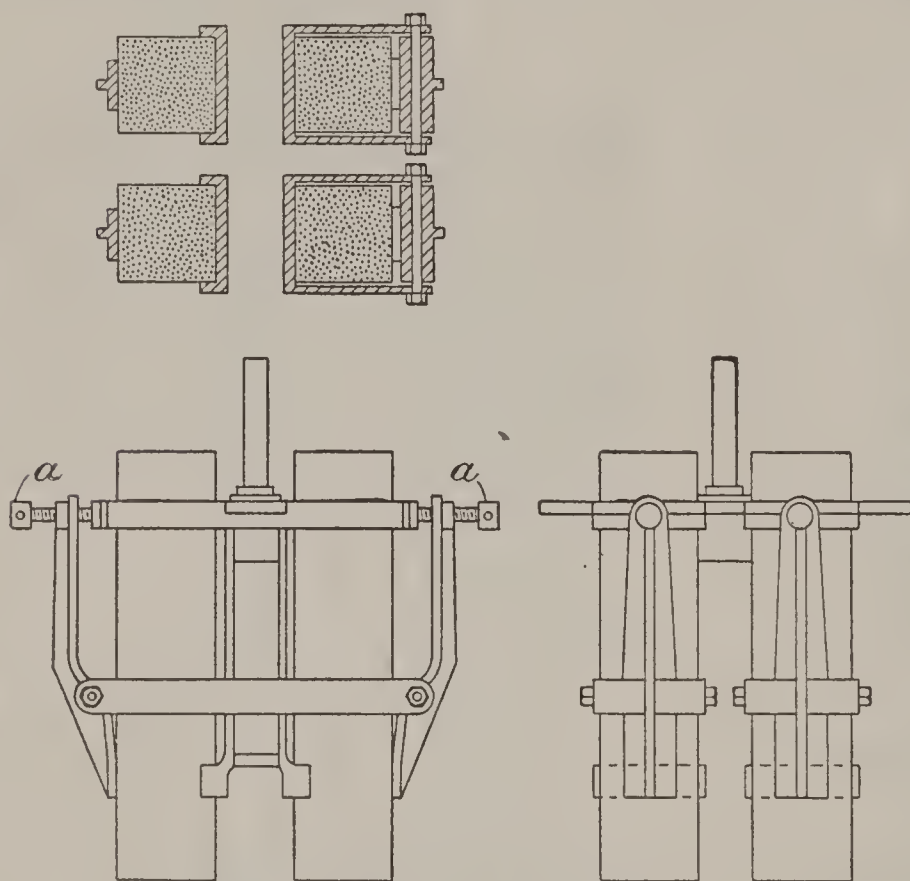


FIGURE 26.—Holder for a block electrode.

of the leaves is determined by the distance of the rings, which make knots when the springs are deflected or stretched.

If the lengths of the single-leaf springs are somewhat different from each other, the elasticity of each spring is used with best efficiency. There is no danger of a short circuit, and the elastic connection is not exposed to the flames of the furnace. This construction has given excellent service, and is used in electric furnaces of various types and employed for a variety of purposes.

ELECTRODE DIMENSIONS AND ELECTRODE BUNDLES.

The best cross-section of electrodes is one of the most important questions for the consumer. Above all, it is determined by the current density of the electrode. Experience is so far the only guide. A number of important papers on this subject have been published or reviewed in Metallurgical and Chemical Engineering. No general formula which would be valid for all cases has yet been found for the calculation of this section. In general, a good electrical and a poor thermal conductivity of the electrode are desirable to reduce the voltage drop and development of heat by the

joulean effect, and so prevent any appreciable loss of heat through the electrodes. Much systematic work remains to be done to find the best conditions for improving the heat balance of furnaces in special cases. In view of the fact that the thermal efficiency of the electric arc furnace is 50 per cent or more, these energy losses through the electrodes are of minor importance.^a

The current density of electrodes depends on the sectional area. It has been proven that round electrodes of from 70 to 100 mm. (2.8 to 4 inches) diameter, as used, for instance, in the Stassano furnace, can carry 20 to 25 amperes per square centimeter (129 to 161 amperes per square inch).^b

Larger sections are used in the electric steel furnaces of Hérault, Girod, Keller, etc. They can not carry any such current density. Up to a section of 400 by 400 mm. (16 by 16 inches) 5 amperes per square centimeter (32 amperes per square inch) are allowed with a temporary overload of 6 to 7.5 amperes per square centimeter (39 to 48 amperes per square inch).

The large differences in the current-carrying capacity of electrodes of different section are partly due to the thermal and electrical conditions mentioned above. Most of the commercial electrodes are made in presses. The capacity of the press is limited by its dimensions and design, and it seems likely that the smaller sections of electrodes produced under a higher pressure have a denser structure. From this it would seem that the current-carrying capacity of the electrode is determined by the pressing capacity of the machines.

The area of surface exposed to the cooling effect and the resistance of the electrode play, no doubt, an important part in these considerations.

It is noteworthy, however, how small the differences in the physical constants of small and large electrodes really are. For the identical carbon mixture we find the following values:

Size of electrode.	Specific resistance.	Specific gravity.
80 mm. ($3\frac{1}{8}$ inches) round.	48 to 50 ohms.	1.55.
300 by 300 mm. ($11\frac{3}{4}$ by $11\frac{3}{4}$ inches) square.	65 to 68 ohms.	1.50 to 1.52.

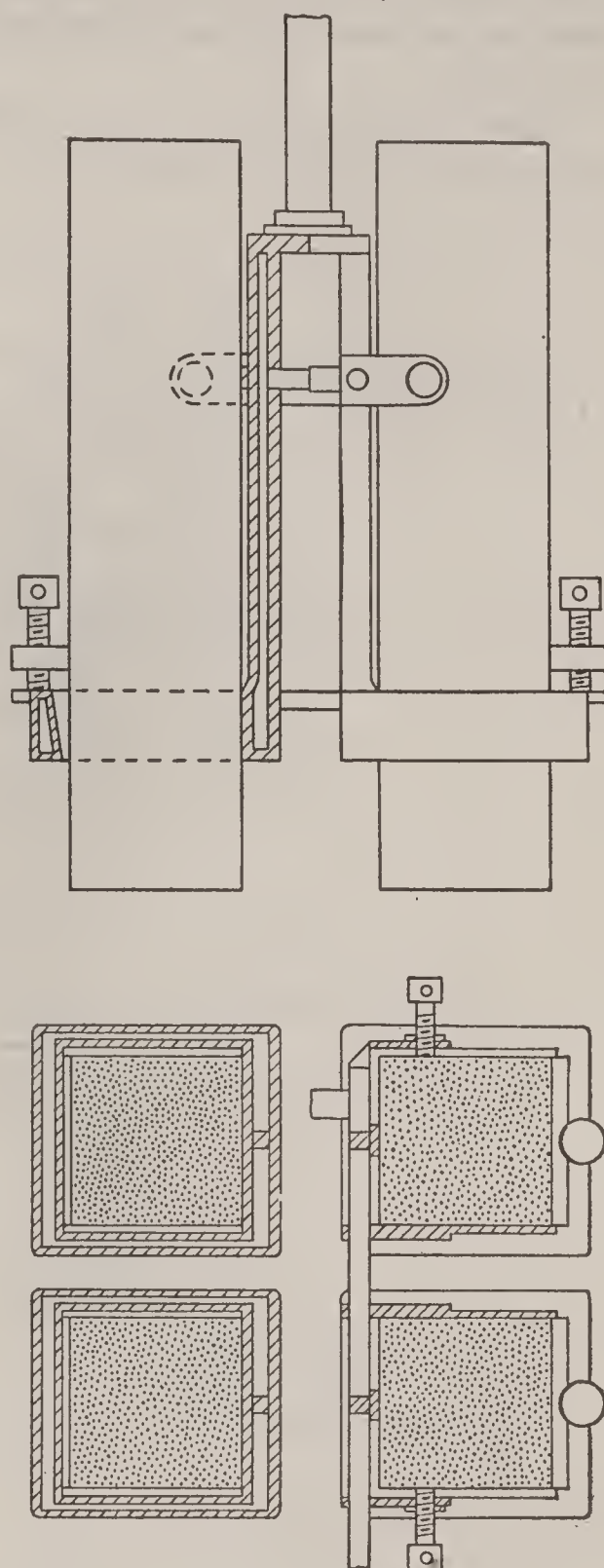


FIGURE 27.—Modification of holder for block electrode.

^a The reviewer of this article states that he differs on this point. The authors of this bulletin also differ with the statement above made.

^b The reviewer states that graphite electrodes are used in these furnaces, and account for the remarkable differences from the figures given immediately after for the electric steel furnaces of Hérault, Girod, Keller, etc.

The consumer of electrodes is anxious to reduce the voltage drop by increasing the section of the electrode. The size of section is limited by the design of the furnace and by the facilities of the electrode manufacturer. Recently much progress has been made so that square electrodes of 550 by 550 mm. (21.6 by 21.6 inches) section and round ones of 625 mm. (25 inches) diameter can now be bought from the leading factories.

These large sections naturally involve some disadvantages. The current-carrying capacity is reduced, the mechanical strength decreases with increasing section, and

the weight of a single electrode reaches a considerable figure. In case of a broken electrode very heavy pieces might have to be removed from a steel bath and an entire heat might be lost by carburization or by cooling off.

The carbide industry, in which large amounts of energy had to be handled at an early date had previously chosen a different way to overcome the above difficulties. Six or eight or other numbers of smaller electrodes were combined into one package or bundle and lowered by means of cranes into the furnace. The size of the individual electrode is often chosen as 250 by 350 mm. (10 by 14 inches). The electrode packages, as illustrated in figures 19, 20, and 45, leave an unused butt.

The amount of these butts is kept at a minimum by the use of proper water-cooled holders, and whatever

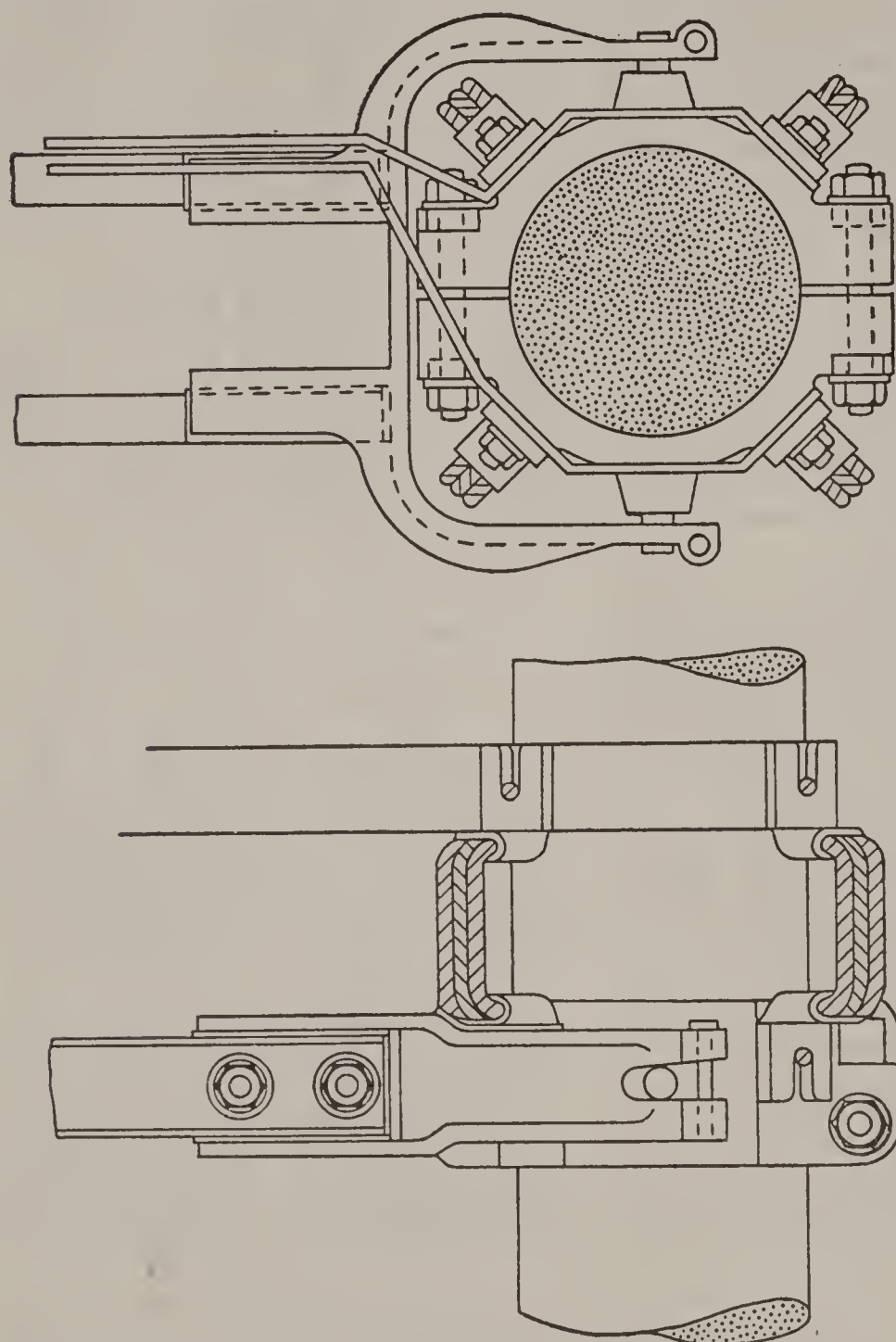


FIGURE 28.—Electrode holder used by the Aktisbolaget Elektrometall Ludwika.

remains is used in the carbide furnaces for patching the hearth.

The electric pig-iron furnace in Dommarfvet used three electrodes, each composed of two individual carbons; the over-all section was 660 by 330 mm. (26 by 13 inches). In Trollhättan ^a each electrode consists of four pieces of 2 mm. (0.79 inch) length and 330 by 330 mm. (13 by 13 inches) section, giving a section of 660 by 660 mm. (26 by 26 inches) over all.

^a This practice is no longer followed at Trollhättan, round electrodes being used.

The combining of smaller pieces offers the additional advantage that the individual electrode has a higher current-carrying capacity and that considerable losses of energy are avoided.

Attempts have been made to increase the conductivity of the electrode, for the sake of decreasing the section, by providing a metallic core. Hérault suggested as

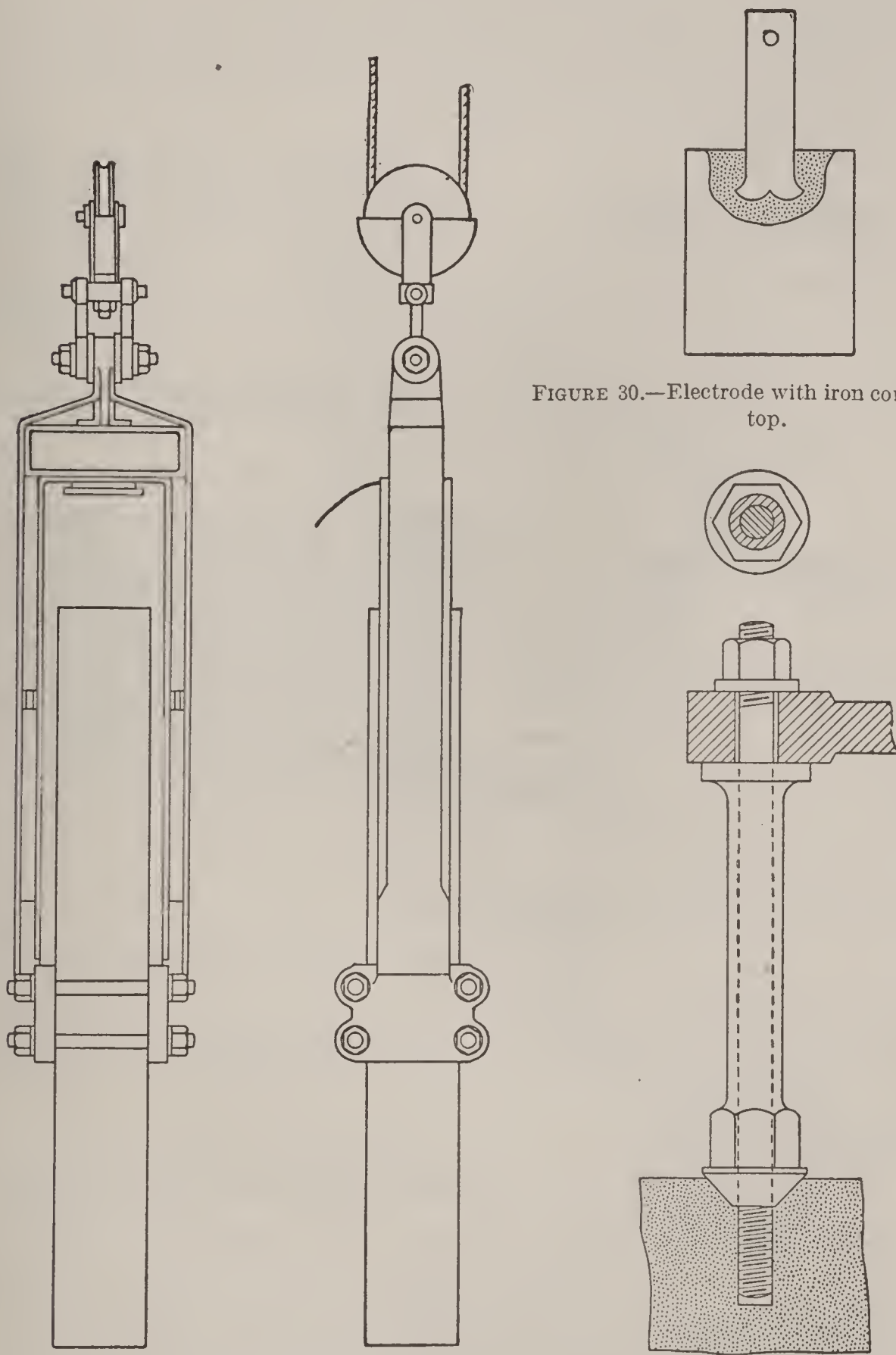


FIGURE 30.—Electrode with iron connection in top.

FIGURE 29.—Electrode holder used by Nathusius. FIGURE 31.—Screw-bolt electrode connection.

early as 1892, in United States Patent 473117, to bore holes through the electrodes and fill these with aluminum alloys or silicon alloys.

The Planiawerke in Rutibor, Germany, improve the conductivity and mechanical strength by metal fillets introduced before or after the baking of the electrodes. (Norwegian Patent 21366, 1910.) The melting or volatilization point of the metal

should be below the reaction temperature of the furnace. Measurements show that the resistance of such electrodes "reinforced" by a metallic core can be reduced to at least one-half of the original with a comparatively small cross section of the core.

The process is particularly intended for use with rather long continuously fed electrodes. Certain alloys can be introduced into the steel by using same as the core.

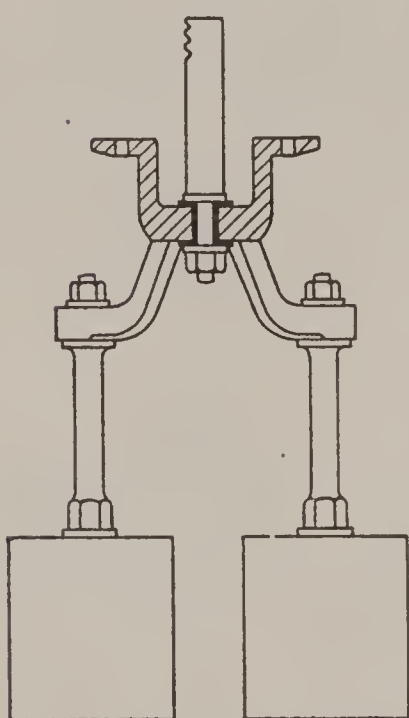
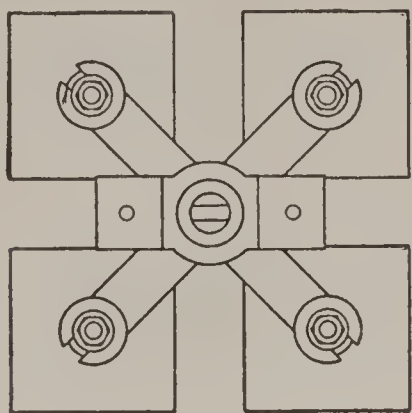


FIGURE 32.—Combination of four screw-bolt electrodes.

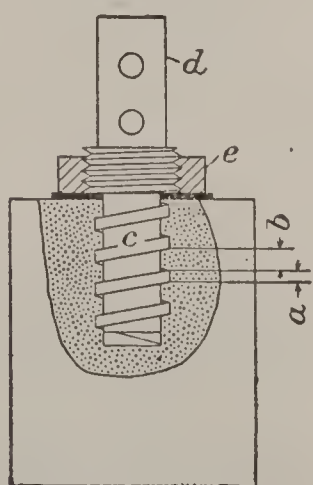


FIGURE 33.—Electrode connection used by Lessing.

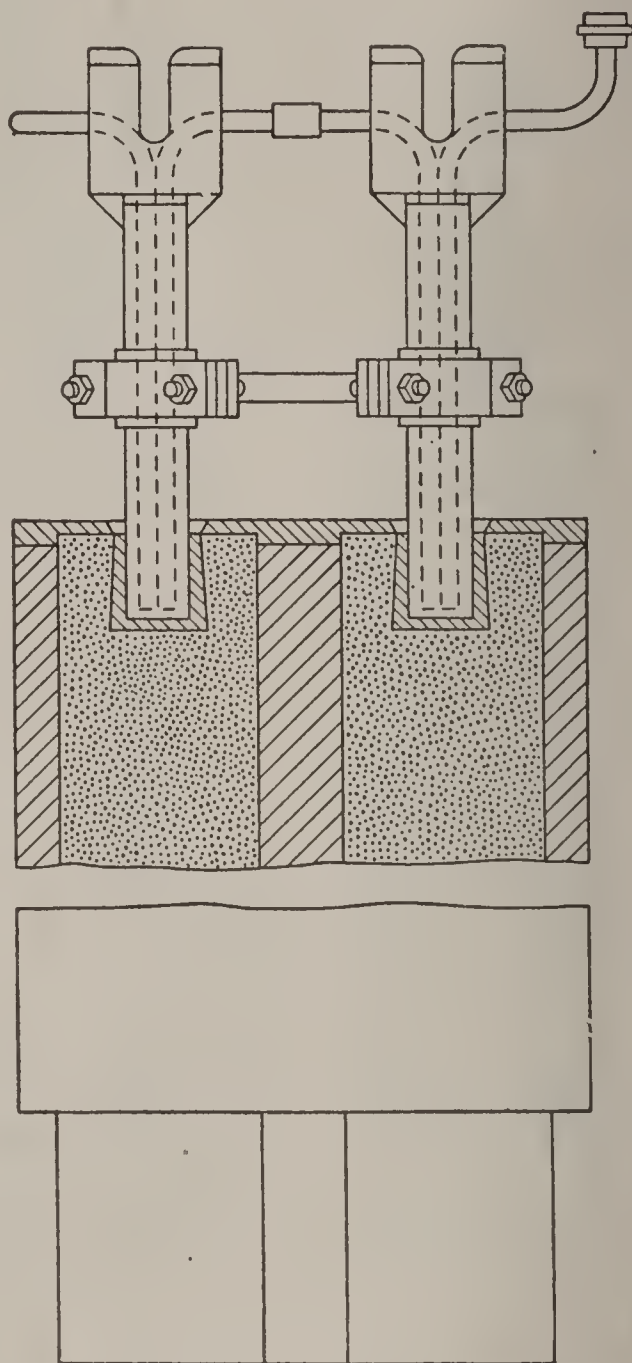


FIGURE 34.—Electrode connection used by Keller.

According to Perkins a tube of the carbon or iron can be filled with lime, iron oxide, or other slag-forming materials and thus deliver the very liquid refining slag at the hottest point of the bath.

Figure 35 shows the form of electrode holder used in the Stassano furnace.

JOINING OF ELECTRODES.

Owing to the fact that the item "cost of electrodes per ton of metal produced" or "per pound of metal produced" is excessive if the electrode is rejected as soon as it becomes too short to permit further use, as is the used electrode shown in figure 15, numerous methods have been devised for joining electrodes together so as to permit continuous feeding. This is generally done in one of three

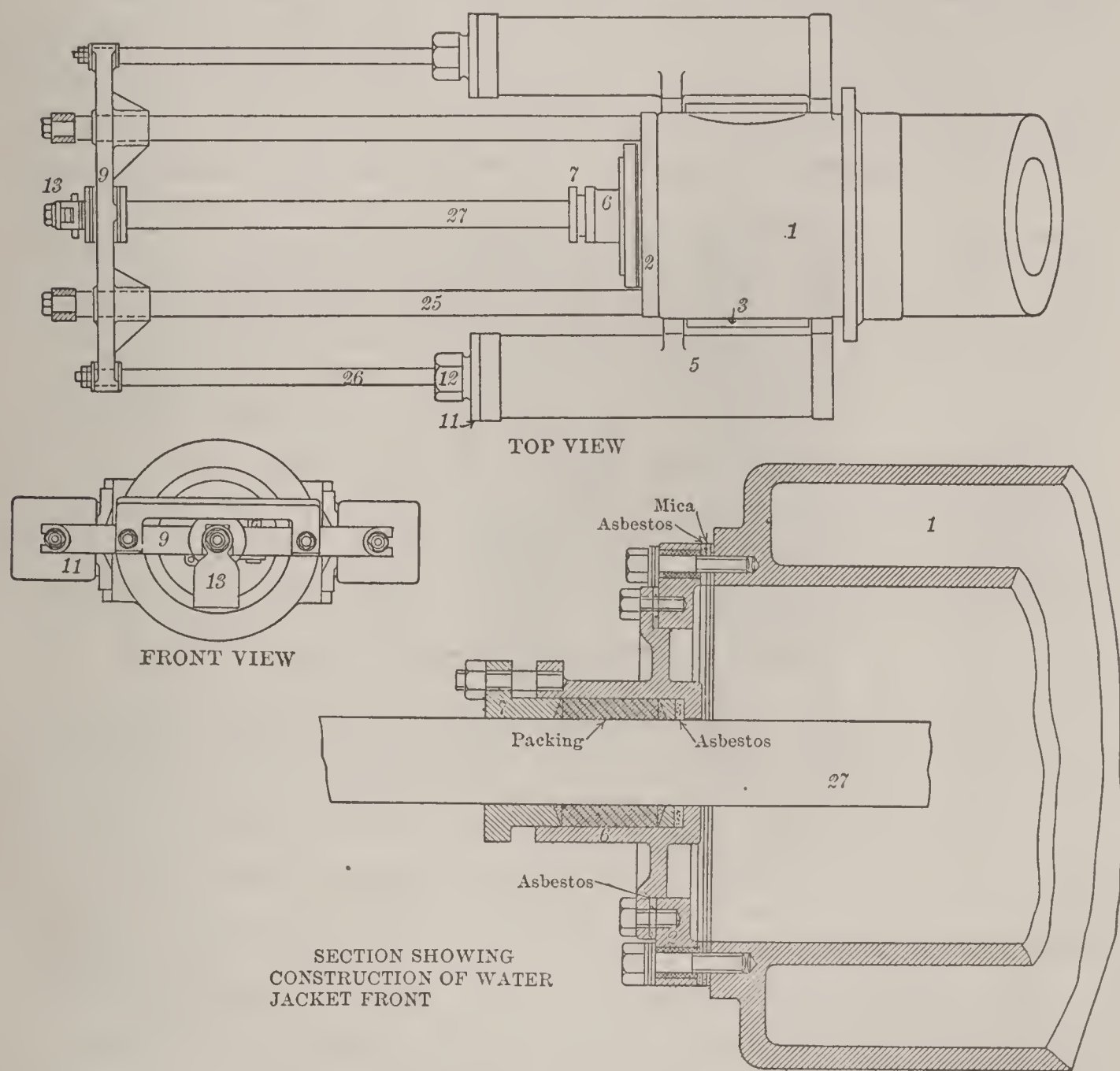


FIGURE 35.—Electrode holder used in Stassano furnace.

ways (fig. 36), namely, (1) by means of a nipple molded with a screw thread, and made out of the same material as the electrode; (2) by means of a cylindrical threaded plug, which is screwed equal distances into the ends of the two electrodes to be joined; or (3) by means of a threaded male and female joint.

Fitzgerald^a states that as a result of numerous experiments it was found that the joint best suited to the average furnace needs is

^a Fitzgerald, F. A. G., Hinkley, A. T., Experiments with furnace electrodes: Trans. Am. Electrochem. Soc., vol. 23, 1913, p. 333.

made by using a threaded plug, in the manner stated second above and that a shallow rounded thread proves better than a pointed one or even a truncated one, as there is less tendency to breakage during assembling.

ELECTRICAL LOSS AT JOINTS.

A certain loss of voltage takes place at these joints. "This loss has been found to be 2 to 2.5 volts for square electrodes of 280 to 330 millimeters (11 to 12 inches) side length with a current density of 5 amperes per square centimeter (32.25 amperes per square inch). This would mean a drop of 4 to 5 volts for an electric steel furnace with two arcs in series, or a loss of 16 to 20 kilowatts for a 500-kilowatt load, which represents 3.5 to 4 per cent of the total energy."^a Such

being the case, the saving in electrode material and time by joining electrodes together has to be balanced against this loss.

The reason for this loss of voltage at the joints is due to the fact that if the electrode be joined by means of a threaded joint, as shown in figure 36, there is liable to be good contact only between the threaded parts (*a* and *c*) of the electrodes, whereas the surfaces of the ends of the

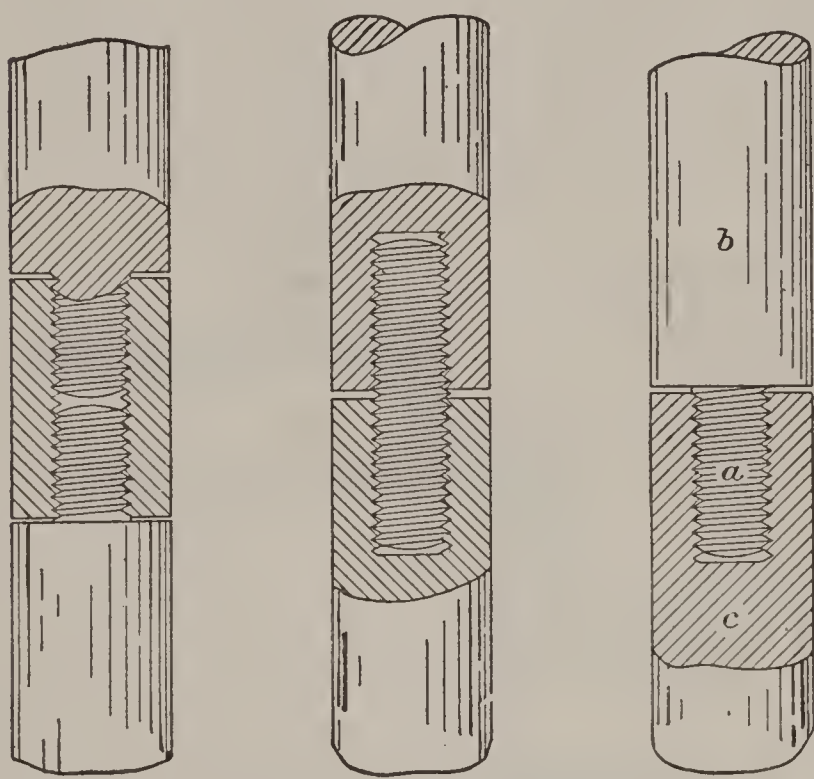


FIGURE 36.—Three types of electrode joints.

electrodes do not come in contact with each other at all. On this account, the current is all crowded on to that part of the electrode carrying the male thread *a*, a condition that not only gives rise to the loss above mentioned, but causes the electrode to become so overheated at this spot as to cause it to go to pieces. It has been suggested that with electrodes of amorphous carbon this difficulty might be overcome by joining the electrodes together by a graphite dowel, as also shown in figure 36. As the conductivity of graphite is about four times that of amorphous carbon, the graphite dowel could be made of such a size as to readily carry the full amount of current carried by the carbon electrode, and in this way the loss above referred to would be reduced to a minimum.

^a Anon., Die Elektrodenfassungen bei Elektroöfen: Stahl und Eisen: vol. 33, 1913, p. 472. Translated by Kennelly, A. E., Met. and Chem. Eng., vol. 11, June, 1913, p. 323.

In this connection the article ^a mentioned above may well be quoted, as follows:

It has also been proposed to make these connecting studs from metal. This, however, would defeat the special purpose of these studs. They would melt and volatilize and not fill their purpose, since their object is to use the last piece of the electrode. This latter aim is completely reached by a joint containing small particles of metal embedded in carbon.

The disadvantages of this method are as follows:

1. The manufacture of the accurate inner thread and nipples and other shapes increases the cost of the electrode construction.

2. The contact resistance causes losses of energy.

3. The joint is apt to loosen and to increase these losses or the end of the electrode in the furnace may drop off, causing short circuits or other troubles; too tight a joint might result in a breaking of the two carbon pieces. (Borchers, *Elektrische Oefen*, 2d ed., p. 143.)

These difficulties are unknown in the use of electrodes which are not joined together; it is mainly a matter of calculation whether to join electrodes or not, and the cost of power, the cost of electrodes, and their length will decide which of the two methods should be used in any case. In cases in which the furnace and the holder allow the use of the electrode down to but a small stump it will be in many cases more advisable not to join the electrodes.

The rate at which electrodes are consumed depends largely upon (1) whether they are exposed to the action of an oxidizing atmosphere, and (2) the nature of the charge, or of the material that is being heated in the furnace.

COST OF ELECTRIC POWER.

Naturally, all other conditions being favorable, the use of the electric furnace in the smelting of ore and the production of metals and ferro-alloys will depend largely upon the cost of electric power.

HYDROELECTRIC POWER.

Ordinarily one thinks of electricity as being most easily developed by water power. In a way this is true, but as to whether power can be as easily and cheaply produced in this manner as by steam or gas will of course depend on the initial cost of installation, which, in turn, depends, among other things, upon the following:

1. The mean average flow of water for 12 months.

2. Whether expensive reservoirs have to be constructed.

3. Whether these reservoirs are situated a long distance from the site of the power plant, thus necessitating the construction of long ditches, flumes, tunnels, or pipe lines.

4. Whether expensive dams have to be constructed.

In all the forms of water power that are available, either one or all of the above-mentioned prerequisites have to be given serious con-

^a Anon., *Die Elektrodenfassungen bei Elektroöfen: Stahl und Eisen*: vol. 33, 1913, p. 472. Translated by Kennelly, A. E., *Met. and Chem. Eng.*, vol. 11, 1913, p. 324.

sideration, although the cost of installation may be as low as \$48 per kilowatt or as high as \$400. As shown in the accompanying table (p. 70), the cost of hydroelectric power to the consumer may vary from \$5 to \$60 per kilowatt-year, depending largely on local conditions.

POWER PRODUCED BY GAS ENGINES AND STEAM TURBINES.

Of late years a great deal has been written regarding the production of cheap electric power by generators driven by gas engines. Although power can be produced cheaply by such generators when waste gases from a blast furnace are utilized, yet if coal be burned in gas producers to produce the necessary gas, even though the coal used be of inferior quality, and by-products are recovered, the cost of electric power made in this way is rather likely to be more than what is generally supposed to be the cost when gas engines are used for driving the generators.

With the introduction of steam turbines of high efficiency the cost of electric power has been much reduced during recent years. From the figures in the two tables following, which were kindly supplied to the Bureau of Mines by a reliable engineering firm, it seems that with large units the cost per kilowatt-hour is less for a turbine-driven generator plant than for a gas-engine plant.

Plants driven by producer gas from bituminous coal and turbine plants of 2,000 kilowatts, 4,000 kilowatts, 6,000 kilowatts, 8,000 kilowatts, and 10,000 kilowatts normal capacity are considered in these tables. In each plant all units are of uniform size. The data are given on the basis of a 75 per cent load factor. The maximum size of the gas engines covered is 2,000 kilowatts; hence corresponding data are given for turbines. Power-plant cost includes everything from real estate up to complete installation of all auxiliaries. The tables follow.

Cost of installation of steam-turbine and of gas-engine power plants.^a

Size of plant.	Size of units.	Number of units.	Cost per kilowatt of installing—	
			Turbine.	Gas engine.
<i>Kilowatts.</i>	<i>Kilowatts.</i>			
2,000	500	4	\$82.00	\$133.00
4,000	1,000	4	71.00	115.00
6,000	2,000	3	67.00	106.00
8,000	2,000	4	64.50	104.00
10,000	2,000	5	62.50	102.50
10,000	{ 6,000	1	61.00
		2		

^a The fixed charges for each type of plant were as follows: Interest, 5 per cent; taxes and insurance, 2 per cent; amortization, 3.5 per cent; total, 10.5 per cent.

Operating cost of gas-engine and of steam-turbine power plants.

COST ITEMS FOR GAS-ENGINE PLANT.^a

Size of plant.	Labor.	Repairs and maintenance.	Oil,waste, and supplies.	Fuel. ^b	Total operating cost.	Fixed charges.	Total.
<i>Kilowatts.</i>							
2,000	0.163	0.044	0.032	0.27	0.509	0.236	0.745
4,000	.145	.039	.029	.27	.483	.204	.687
6,000	.130	.035	.026	.27	.461	.188	.649
8,000	.124	.034	.025	.27	.453	.184	.637
10,000	.123	.033	.025	.27	.451	.182	.633

COST ITEMS FOR TURBINE PLANT.^a

2,000	0.106	0.019	0.020	0.41	0.555	0.133	0.688
4,000	.088	.017	.017	.36	.482	.115	.597
6,000	.072	.017	.014	.338	.441	.108	.549
8,000	.067	.016	.013	.332	.428	.104	.532
^c 10,000	.066	.016	.013	.33	.425	.101	.526
^d 10,000	.064	.0155	.0125	.323	.415	.099	.514

^a In cents per kilowatt-hour.
^b At \$3 a ton.
^c Consisting of five 2,000-kilowatt units.
^d Consisting of one 6,000-kilowatt unit and two 2,000-kilowatt units.

In considering items pertaining to the two 10,000-kilowatt turbine plants, it should be noticed that the plant with the larger units has lower costs for each item. A similar arrangement might be made for the other turbine plants, with a decreased cost resulting; for example, the plants might be arranged as follows:

2,000-kilowatt plant in one 1,000-kilowatt unit and two 500-kilowatt units.

4,000-kilowatt plant in one 2,000-kilowatt unit and two 1,000-kilowatt units.

6,000-kilowatt plant in one 3,000-kilowatt, one 2,000-kilowatt, and one 1,000-kilowatt unit.

8,000-kilowatt plant in one 4,000-kilowatt unit and two 2,000-kilowatt units.

If the heating value of the coal is assumed to be 13,000 British thermal units per pound for all prices of coal, the only item in the operating cost affected by the varying price would be fuel cost. For example, in a 4,000-kilowatt turbine plant the effect of varying fuel prices might be as indicated below:

Cost of coal per ton.	Fuel cost per kilowatt-hour.
\$1	\$0.12
2.....	.24
3.....	.36
4.....	.48

In the table following are given some approximate cost figures for hydroelectric power when used for electric furnaces or electrochemical purposes, with a high load factor:

Cost of electric power for electric furnaces or electrochemical purposes with a high-load factor.

Location of plant.	Power company.	Nature of power.	Cost per kilowatt-year.	Product at the present time.
Braintree, England.....	Lakes & Elliott Steel Foundry.	Gas engine.....	\$110	Steel.
Sheffield, England.....	Municipal System.....	Steam.....	87	Do.
La Praz, Savoie, France..	Electrometallurgical Co. of France.	Hydroelectric.....	16	Aluminum, steel, carbide.
Livet, Isère, France.....	Keller, Leleux & Co.....do.....	13	Ferro-alloy.
Ugine, Savoie, France....	Girod Electrometallurgical Co.do.....	13	Ferro-alloy, steel.
Bonn, Germany.....	Municipal System.....	Steam.....	92	Steel.
Remscheid, Germany.....	Lindenber Steel Co.....do.....	87	Do.
Völklingen, Germany....	Röchling Iron & Steel Co..	Steam, gas.....	83	Do.
Meraker, Norway.....	Meraker Electric Smelting Co.	Hydroelectric.....	8	Ferro-alloys, carbide.
Notodden, Norway.....	Norwegian Saltpeter Co...do.....	7	Nitrates.
Rjukan, Norway.....do.....do.....	5	Do.
Tyssedal, Hardanger, Norway.	Tyssedal Power Co.....do.....	10	Carbide, cyanamide.
Trollhättan, Sweden.....	Royal Power Plant.....do.....	9 to 12	Iron, zinc, ferro-alloy, paper.
South Verron, Vt.....	Connecticut River Power Co.	Water and steam..	53	General.
Shelburne Falls, Massdo.....	Water (secondary)	29	Do.
Turners Falls, Mass.....	Turner Falls Power Co...	Water and steam .	60	Do.
Do.....do.....	Water (secondary)	26	Do.
Rumford Falls, Me.....	Rumford Falls Power Co..	Hydroelectric.....	26	Paper.
Niagara Falls, U. S.....do.....do.....	26	Electrochemical.
Niagara Falls, Canada...do.....do.....	20	Do.
South Carolina.....	Southern Power Co.....do.....	25	Nitrates.
South Chicago, Ill.....	Illinois Steel Co.....	Steam, gas.....	43	Steel.
South Carolina.....	Southern Aluminum Co...	Hydroelectric.....	25	Aluminum.
SNAKE River, Idaho.....	Southern Idaho Power Co.do.....	12	General.
Heroult, Cal.....	Northern California Power Co.do.....	16	Pig iron.

OPERATING CONDITIONS OF ELECTRIC FURNACES.

In the operation of an electric furnace one or more of the following conditions may be encountered:

A high enough temperature may not be reached in consequence of one of the following conditions:

- 1. The furnace may be “underpowered,” that is, may not have enough electric current supplied to it, and this in turn may be due to lack of either power-line supply, transformer capacity, or insufficient voltage.
- 2. The material to be treated may be fed too rapidly.
- 3. The radiation losses may be too great.

Then, too, the temperature may be too high. With furnaces using voltage control this fault may be remedied by cutting down the voltage of the current used, but if a constant voltage be used and the temperature is too high about the only remedy is to feed in more charge. If this procedure will remedy the defect it indicates that the furnace has a capacity greater than that which has been

calculated for it. However, an "overpowered" furnace is generally short lived, as the walls are unable to conduct the heat away fast enough to prevent their destruction through fusion. Then, too, as is self-evident, the higher the temperature the greater will be the heat losses through the furnace walls and through the electrodes. Hence, if there is no advantage to be gained from a temperature higher than is needed for the necessary reactions of the process, it can be said that if in operating an electric furnace an excessive temperature is obtained an error has been made in calculating the size of the furnace or in the amount of energy needed. However, as is also self-evident if the excess heat can be absorbed by a faster feeding of the material to be smelted the heat losses per pound of material smelted will be less, for the heat losses through the electrodes and walls are continuous, and hence are less per pound of material smelted if 20 pounds be treated per minute than if only 10 pounds be treated.

CONCLUSION.

In conclusion the authors repeat that their object has been not to present a technical discussion of the electric furnace, but to acquaint the layman, so to speak, with some of the most important essentials in connection with the use and construction of the electric furnace as used in metallurgical work and to assist those who are unfamiliar with the subject to a better understanding of the processes described in parts II and III of this bulletin and in Bulletin 67 on "Electric Furnaces for Making Iron and Steel."

PART II. THE SMELTING OF METALS IN THE ELECTRIC FURNACE.

By DORSEY A. LYON and ROBERT M. KEENEY.

INTRODUCTION.

The object of Part II of this bulletin is to present briefly a statement of the extent to which the electric furnace has been applied to the metallurgy of metals when the product is in the form of the metal and not of a ferro-alloy. For example, the metals copper and zinc are discussed in this part and the ferro-alloys, such as ferrochrome and ferrotungsten, are discussed in Part III. No attempt is made to present exhaustively the details of each particular process, as persons desiring such information can obtain it from the references given.

Under this part there is discussed the electric smelting of the ores of aluminum, copper, gold and silver, iron, lead, complex sulphides, and zinc. The subject of iron has been treated briefly, as compared with its importance, because it has already been discussed fully in Bulletin 67.^a

ALUMINUM.

INTRODUCTION.

In the manufacture of aluminum the electric furnace has a field entirely its own, for there is no process by which alumina is reduced to aluminum in a combustion furnace. In the metallurgy of aluminum, as stated in Part I of this bulletin, the electric current not only supplies the necessary heat, but also reduces the metal from the ore. The first application of electrolysis to the reduction of aluminum was in 1864 when Bunsen and Deville, each working independently, obtained the metal by the electrolysis of fused aluminum chloride.^b Previous to the introduction of the present method the aluminum was obtained chiefly through the reduction of the halide salts of the metal with metallic sodium.

In 1884 the Cowles process was patented, in which alumina was reduced with carbon in the presence of copper in an electric furnace of the resistance type. An alloy^c of aluminum and copper was

^a Lyon, D. A., and Keeney, R. M., Electric furnaces for making iron and steel: Bull. 67, Bureau of Mines, 1913, 142 pp., 36 figs.

^b Thompson, M. D., Applied electrochemistry, 1911, p. 228.

^c U. S. patent 319795.

the product. It was known then that alumina could be reduced to aluminum as a result of being heated with carbon to a temperature higher than $2,100^{\circ}\text{C.}$, but all of the aluminum formed was mixed with aluminum carbide, from which it was removed by a refining process. In the Cowles process the copper was used to prevent the formation of aluminum carbide, but of course the process had the objection that the aluminum was not obtained as pure metal but as an alloy.

In 1886 Hall^a found that alumina dissolved in a molten mixture of aluminum fluoride and the fluoride of another metal formed an electrolyte which could be decomposed by the electric current according to the reaction $\text{Al}_2\text{O}_3 = 2\text{Al} + 3\text{O}$. About the same time that Hall made his discovery, Héroult in France made a similar independent discovery.

PREREQUISITES FOR THE PRODUCTION OF ALUMINUM BY PRESENT METHODS.

Although aluminum is one of the most abundant of the elements that make up the earth's crust, nevertheless those of its ores that are suited to the process of aluminum manufacture, as carried on at present, are comparatively scarce. About the only one now used for this purpose is bauxite. The great bulk of that produced in the United States is mined in Arkansas and Tennessee. Phalen^b states that some bauxite is now mined in central Georgia.

PROCESSES FOR PURIFYING BAUXITE.

As all the impurities in the alumina pass into the metal produced, it is necessary in both the Hall and the Héroult processes to use alumina as pure as possible. This is why the Bayer process found considerable application both here and abroad for the purification of bauxite. In this process the bauxite is fused with the carbonate or sulphate of sodium and the fused mass is washed with water; then the sodium aluminate obtained in solution is decomposed by adding aluminum hydrate and constantly stirring. Richards^c states that the decomposition of the solution goes on until the quantity of alumina remaining in solution is to the sodium protoxide as 1 to 6. This precipitation takes place in the cold, and the pulverent aluminum hydrate separated out is easily soluble in acids. The alkaline solution remaining is concentrated by evaporation, taken up by ground bauxite, dried, calcined, melted, and thus goes through the process again.

^a U. S. patents 400664 and 400666.

^b Phalen, W. C., Bauxite and aluminum: Mineral Resources U. S. for 1911, U. S. Geol. Survey, 1912, p. 928.

^c Richards, J. W., Aluminum, its properties, metallurgy, and alloys, 3d ed., 1896, p. 144.

Hall also invented and patented at least two processes for this purpose, including the lime process of wet chemical treatment (U. S. patent 663167, Dec. 4, 1900), which was installed at East St. Louis, and a process whereby the bauxite is mixed with a small percentage of carbon and calcined, and after calcination is again mixed with about 8 to 10 per cent of carbon, some ferric oxide, possibly some sort of a flux being added. The impurities such as iron and silicon are reduced, and form ferrosilicon of impure grade, which sinks to the bottom of the electric furnace, leaving the purified alumina at the top. After the alumina has cooled, it is separated from the ferrosilicon by electromagnetic methods if necessary. In the patent specification, it is stated that aluminum powder is also added, but this is not now thought necessary. In England, the British Aluminum Co. (Ltd.) purify bauxite by dissolving it in caustic soda and then precipitating the alumina by the addition of some previously precipitated alumina.^a

HALL PROCESS FOR PRODUCING ALUMINUM.

Although more than 20,000 tons of aluminum is produced each year in the electric furnace, little published information is available concerning the methods of manufacture, as all manufacturers try to keep their processes secret. In this country practically all of the aluminum produced is obtained by the use of the Hall process. The essential facts regarding this process are as follows: The ore (bauxite) is first purified by being treated either by the lime process or the electric-furnace process. It is then charged into an electric furnace containing a bath of cryolite or a mixture of aluminum fluoride with the fluoride of another metal, the bath being kept molten by the heat generated by the passage of the electric current. The alumina is thus decomposed into aluminum and oxygen by the electrolyzing action of the direct current. A concise description of the process used in practice is given by Neumann and Olsen^b from whose article the following information is quoted:

A photograph of an aluminum cell has been published only once, so far as is known, by J. W. Richards;^c the cell shown is evidently a cell of Charles M. Hall, as used some years ago. The illustration shows 20 round anodes in two parallel rows in a high iron box. Quite detailed notes illustrated by sectional diagrams were also given later by Winteler.^d The cells described by Winteler are also rectangular, and are also provided with a double row of carbon electrodes. More recently, circular cells have been introduced, the anodes being distributed over the whole surface.

^a Cachu, William, London Elec. Rev., 1911, vol. 68, p. 99.

^b Neumann, B., and Olsen, H., Production of aluminum as a laboratory experiment: Met. and Chem. Eng., vol. 7, 1910, p. 185.

^c Electrochemical Industry, vol. 1, 1903, p. 160.

^d Winteler, Aluminium Industrie, 1903.

HÉROULT PROCESS FOR THE MANUFACTURE OF ALUMINUM.

In principle, the Héroult process is the same as the Hall process, the reported differences being in the composition of the electrolyte and possibly in the shape of the furnace. In fact, the two processes are so similar that when the discovery of the reaction was made simultaneously, there was no patent litigation, but the Hall Company took the American field and the Héroult people the European, a condition that was maintained until recently, when the last of the Hall and allied patents expired in the United States. Now the Héroult interests are erecting a plant in North Carolina.

In Europe at the present time the manufacture of aluminum is conducted exclusively in furnaces (see fig. 37) having a carbon block as the cathode, forming the bottom of the furnace, upon which the molten aluminum collects, and several upper carbon anodes, while the bath is kept molten entirely by the heat generated due to the passage of the electric current. The cross section is in all cases rectangular. The furnaces seen recently by one of the writers at a French aluminum plant were rectangular and of the general shape of the Héroult steel furnace, but not so large. They

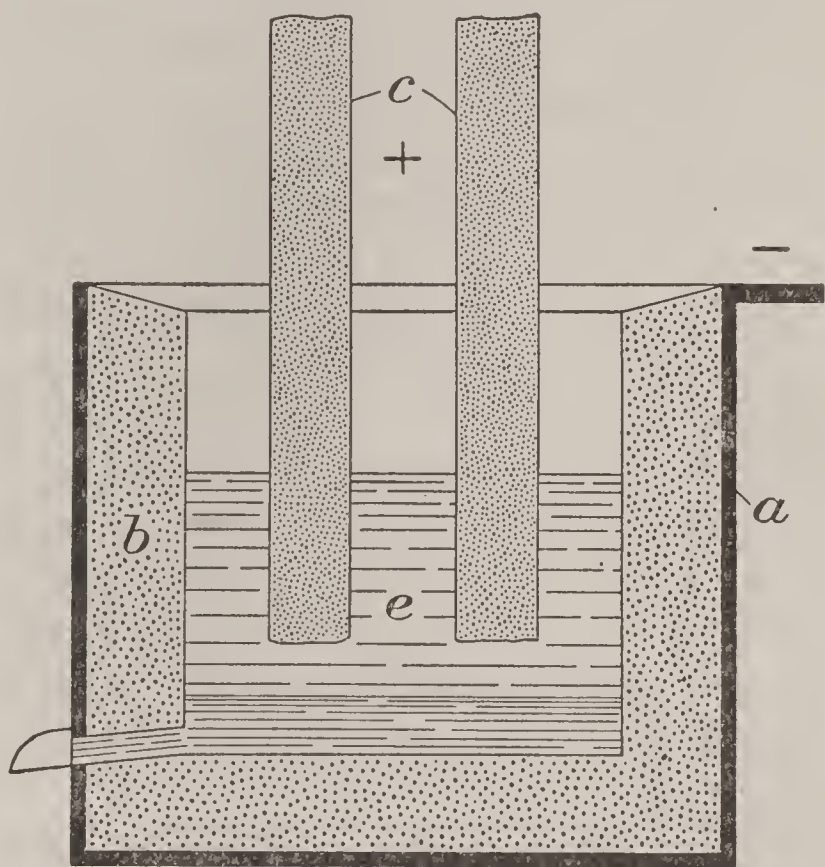


FIGURE 37.—Aluminum furnace, *a*, shell; *b*, lining; *c*, anodes; *e*, bath.

were about 5 feet long, 3 feet wide, and 3 feet deep externally. Each furnace rested on the floor of the building without special foundations, so that furnaces could be picked up and moved around readily with a crane. There were eight carbon anodes projecting into the bath and connected to the positive side of the circuit, while the electrodes in the carbon bottom are connected to the negative side of the circuit. There is an electrode in the bottom corresponding to each of the anodes. In this case the bottom is only partially covered by the molten aluminum.

A second type of furnace has a bottom composed of a mixture of tar and ground coke or electrode waste, which is rammed in around four large iron plates connected to the negative side of the circuit. There are four rows of anodes containing six to eight electrodes per

row. In both types of furnace the bottom is inclined toward the tap hole, and each electrode is arranged so that it can be regulated independently of the others.

In the Héroult as in the Hall process the composition of the bath is kept secret, but in both processes the specific gravity of the bath must be less than that of the metal, so that the latter can collect on the bottom. Cryolite melts at $1,000^{\circ}\text{C}$., but its melting point is reduced to 915°C . by the addition of 5 per cent alumina. By the use of calcium fluoride the melting point of cryolite can be further reduced to 800°C . Cryolite has a specific gravity of 2.92 when solid and 2.08 when liquid, whereas aluminum has a specific gravity of 2.6 when solid and 2.54 when liquid. It is not considered advisable to have more than 25 per cent of aluminum, in the form of alumina, in the molten bath. Although aluminum fluoride can also be used for lowering the melting point, the most common flux is calcium fluoride.

Theoretically, 42 grams (0.092 pound) of aluminum per kilowatt-hour should be produced by the electrolysis of alumina in a bath of cryolite. Practically, the foreign manufacturers using the Héroult process make 30 grams (0.066 pound) of aluminum per kilowatt-hour, which gives an efficiency of 71 per cent.

The furnaces take about 75 kilowatts each and produce from 50 kilograms (110 pounds) to 55 kilograms (121 pounds) per 24 hours. The voltage used is from 8 to 10 volts and the density of current from 1.5 to 3 amperes per square centimeter. The consumption of alumina per pound of finished aluminum is theoretically 1.888 pounds and practically 2 pounds. For each pound of aluminum produced there is a consumption of 0.1 pound of cryolite and 0.9 pound of carbon anodes.

The main progress in aluminum manufacture, as far as can be determined, has been in the purification of the bauxite into alumina before it is charged into the electrolytic cell, mention of which has already been made.

SERPEK PROCESS.

A new process, the Serpek, has been devised and is being introduced in Europe. In this process bauxite is heated to a temperature sufficient to cause the formation of aluminum nitride, which is decomposed by the action of water into alumina and ammonia, according to the reaction $2\text{AlN} + 3\text{H}_2\text{O} = \text{Al}_2\text{O}_3 + 2\text{NH}_3$. For the preliminary reaction an electric furnace is not absolutely necessary, as producer gas can be used as a fuel, but of course an electric furnace is used to reduce the refined alumina to metallic aluminum. The process thus provides a method for the fixation of nitrogen and for the purification of bauxite to produce pure alumina for the electrolytic cell.

The raw bauxite is first calcined by being passed through a revolving cylindrical kiln similar to a cement kiln. The calcined bauxite passes by gravity to another similar kiln, before entering which it is mixed with carbon and then treated with nitrogen. Halfway down the lower kiln is a detachable electric resistance furnace which is intended to raise the temperature to $1,800^{\circ}$ to $1,900^{\circ}$ C. The material is discharged from the lower end of this kiln into an air-tight receiver. Producer gas, consisting of one-third carbon monoxide and two-thirds nitrogen, is introduced at the lower end of the lower kiln. This gas enters at a temperature of 400° C., but when it reaches the electrically heated part of the furnace and attains a temperature of $1,800^{\circ}$ C., it reacts upon the charge to form aluminum nitride.

PRESENT STATUS OF ALUMINUM MANUFACTURE IN THE UNITED STATES.

The following table,^a showing the production of aluminum since the beginning of the industry in 1883, and the consumption of the metal since 1904, gives an idea of the magnitude of the industry in this country:

Production of aluminum in the United States, 1883-1904.

Year.	Pounds.	Year.	Pounds.
1883.....	83	1899.....	6, 500, 000
1884.....	150	1900.....	7, 150, 000
1885.....	283	1901.....	7, 150, 000
1886.....	3, 000	1902.....	7, 300, 000
1887.....	18, 000	1903.....	7, 500, 000
1888.....	19, 000	1904.....	^b 8, 600, 000
1889.....	47, 468	1905.....	^b 11, 347, 000
1890.....	61, 281	1906.....	^b 14, 910, 000
1891.....	150, 000	1907.....	^b 17, 211, 000
1892.....	259, 885	1908.....	^b 11, 152, 000
1893.....	333, 629	1909.....	^b 34, 210, 000
1894.....	550, 000	1910.....	^b 47, 734, 000
1895.....	920, 000	1911.....	^b 46, 125, 000
1896.....	1, 300, 000		
1897.....	4, 000, 000	Total.....	239, 751, 779
1898.....	5, 200, 000		

Until recently there was only one company producing aluminum in the United States, and that company controlled the Hall patents. Now that these patents have expired, as before stated, French interests are erecting a plant at Whitney on the Yadkin River in North Carolina. Work is being pushed rapidly on this plant, which is to be capable of producing 5,000 tons annually.

In 1907, the company controlling the Hall patents had a total plant capacity of about 57,000 kilowatts, of which 45,000 was in the United

^a Phalen, W. C., Bauxite and aluminum: Mineral Resources U. S. for 1911, U. S. Geol. Survey, 1912, p. 932.

^b Consumption.

States and 12,000 in Canada. The company failed to get permission from Congress to increase some of its water-power developments and at present about 4,000,000 pounds of aluminum is annually imported into the United States, as this company alone can not supply the domestic demand. It is, however, building a plant in Tennessee that will increase its productive capacity considerably. The demand for aluminum has increased rapidly during the past few years, and as there are no longer any patent restrictions, it is probable that other companies will begin manufacture in the United States.

FACTORS GOVERNING GROWTH OF ALUMINUM INDUSTRY.

As can be readily seen from what has already been stated on this subject, the growth of the aluminum industry will depend, among other things, upon the discovery of new ores of aluminum; that is, ores that can be used in the production of aluminum by the processes now employed for that purpose, or upon the development of processes for the treatment of ores that are not now suited to the Héroult or Hall processes, or to processes similar to them.

NEW SOURCES OF ALUMINA.

As before stated, bauxite is practically the only ore now used for the production of aluminum. However, Phalen^a states that experiments with alunite, deposits of which occur near Marysvale, Utah, and in Colorado, California, Arizona, and Nevada, indicate that it may at some future time become an important source of alumina.

The mineral alunite is a hydrous sulphate of aluminum and potassium, having the formula $K_2O \cdot 3Al_2O_3 \cdot 4SO_3 \cdot 6H_2O$, and according to laboratory experiments made by Schaller of the Geological Survey contains, when ignited, 32.7 per cent of available potassium sulphate, which can be extracted by simple water leaching and evaporation, thus leaving a residue of 67.3 per cent, which consists of nearly pure aluminum oxide. Such being the case, it would at first thought seem as if this ore would be valuable both for its potash content and for the aluminum that it contains. As a matter of fact, as pointed out in the report of the Geological Survey, the mineral has attracted attention as a possible source of potash. However, as Phalen states, present known deposits are so far from Eastern markets that alunite ores may not be able to compete with Eastern bauxite ores. But as bauxite contains a considerable proportion of iron oxide, silica, and titanium oxide, and requires purifying before being reduced to metallic aluminum, whereas the residue remaining after the leaching of ignited alunite consists of practically pure aluminum oxide, it would seem that this advantage, together with the fact that the

^a Phalen, W. C., Bauxite and aluminum: Mineral Resources U. S. for 1911, U. S. Geol. Survey, 1912, pp. 424-426.

available potassium sulphate that alunite contains is a valuable by-product, would more than offset added transportation charges to Eastern markets.

PROCESSES FOR PRODUCING ALUMINUM FROM ORES OTHER THAN OXIDES.

Patents covering the production of aluminum from molten electrolytes seem to cover every possible method of obtaining aluminum from its ores,^a so that one would think that little more could be done along this line; but, as in other branches of metallurgical work, new ideas are being constantly advanced and patents are obtained on new and generally untried inventions until they seem to be without end.

As previously stated, bauxite, which is practically the only ore used for the production of aluminum, is found in comparatively few places. Moreover, it not only has to be transported long distances, but also requires a costly preliminary treatment for the purpose of freeing it as far as possible from all constituents other than alumina. Aluminum silicates, on the other hand, can be had almost anywhere, and it would therefore seem that the logical thing to do, as regards discovering new sources from which aluminum can be obtained, would be to perfect commercially feasible processes for the production of the metal from those aluminum minerals that are so common. Considerable experimental work of this sort has already been done and several patents have been granted on processes for extracting aluminum from ores containing it in some form other than the oxide.

TONE'S METHOD.

The method proposed by Tone^b consists of two steps. In the first step the aluminum-bearing ore is mixed with the proper quantity of carbon, and the aluminum is reduced to aluminum carbide. The aluminum carbide is then mixed with some other ore—silica, for example. The aluminum carbide and the silica are to be in the proportions expressed by the equation $2\text{Al}_4\text{C}_3 + 3\text{SiO}_2 = 8\text{Al} \cdot 3\text{Si} + 6\text{CO}$., and the mixture is fused in an electric furnace yielding, as shown by the above equation, a silicon-aluminum alloy.

If metallic aluminum be desired, the aluminum carbide is mixed with aluminum oxide in the proportion expressed by the chemical equation $\text{Al}_4\text{C}_3 + \text{Al}_2\text{O}_3 = 6\text{Al} + 3\text{CO}$. As has been shown by Hutton and Petavel,^c although the oxide of aluminum (alumina) can be

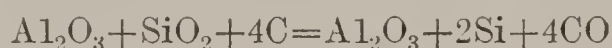
^a See digest of patents taken out prior to July, 1902, in *Electrochemical Industry*, vol. 2, 1904, pp. 165, 166, 210.

^b U. S. Patent 961913, June 21, 1910.

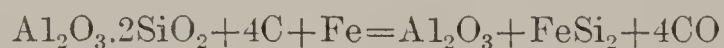
^c Hutton, R. S., and Petavel, J. E. On the direct reduction of alumina by carbon: *Electrochem. and Met. Ind.*, vol. 6, 1908, p. 104.

readily reduced to the metal by carbon, yet the temperature at which the reduction takes place is sufficiently high to volatilize the metal. For this reason, if for no other (as, for example, the reduction of other elements contained in the ore, which would alloy with the aluminum), Tone claims that it is better to divide the process into two parts, thus entailing little or no volatilization loss, and that the production of carbides with the electric furnace is highly efficient. However, as will be noted, the use of aluminum oxide is an essential part of the process where the production of metallic aluminum is desired.

Tone has also taken out two other patents on the production of aluminum from silicates. In one of these (No. 906,338) kaolin is mixed with carbon for the purpose of reducing the silica but not the alumina. The mixture is fused in an electric furnace, the products being alumina and silicon as shown by the following equation:



The alumina after separation from the silicon can be used for the production of aluminum by the usual methods. Tone states that the reaction shown above may be facilitated by adding base metals or ore to the charge, as the base metal alloys with the silicon and so brings about a more complete reduction of the silicon. For example, if 222 parts of calcined kaolin be mixed with 48 parts of carbon and 56 parts of iron, then the reaction in the electric furnace is shown by the following equation:



After the fused product has been tapped from the furnace, the silicide is separated from the alumina, after which the alumina may be treated for the production of aluminum in the usual manner.

Alf-Sinding Larsen has also taken out a patent (U. S. No. 927758) that closely resembles that of Tone. In his process he proposes to fuse silicates of aluminum in an electric furnace in contact with iron (scrap iron or iron ore) and coke. The product of the fusion is ferro-silicon and alumina. The alumina can then be treated in the usual manner for the production of aluminum.

BETTS' PROCESS.

Briefly stated, the process proposed by Betts consists of the following steps:

1. Iron ore and aluminum ore such as bauxite or kaolin are charged with fuel into a blast furnace, similar to an iron blast furnace, but operated at a higher temperature than the latter. The product is iron-aluminum-silicon.

2. As aluminum is the most readily oxidized, sulphurized, or chloridized metal of the iron-aluminum-silicon compound, aluminum may

be extracted by means of either oxidizing, sulphurizing, or chloridizing agents. For example, by treating the iron-aluminum-silicon metal with silica at a high temperature, the aluminum of the metal would be oxidized by the oxygen of the silica, and there would be formed pure aluminum oxide and ferrosilicon, provided the iron ore is free from lime and magnesia.

3. Betts claims that aluminum sulphide can be more readily and cheaply reduced to aluminum than can the oxide (alumina), and so instead of reducing the alumina to metallic aluminum, as is done in the Hall and Héroult processes, he proposes to treat it with iron sulphide, thus forming aluminum sulphide. This part of the process however, is carried out in two steps. The alloy is first treated with an insufficient quantity of iron sulphide to react with the aluminum present, resulting in practically pure aluminum sulphide. This aluminum sulphide is then reduced by electrolysis to aluminum and sulphur.

As the alloy remaining after the treatment in the second step by fusion with silica still contains considerable aluminum, it is further treated with an excess of iron sulphide. The result of this treatment is a matte, which is then used alone or with additional fresh iron sulphide in the next treatment of the aluminum alloy.

IRON.

PRESENT STATUS OF ELECTRIC FURNACE IN SMELTING IRON ORES.

USE OF COKE AND CHARCOAL.

As is well known, of the coke charged into an iron blast furnace, only about two-thirds is used for producing the heat necessary for carrying on the process, the other one-third being used as a reducing agent. Therefore, if the electric furnace is used for the smelting of iron ores, only enough carbon has to be supplied to unite with the oxygen of the ore, or in other words, to reduce it, and for this reason the smelting of iron ores in the electric furnace is of much importance to the Western Coast States, for there are found iron-ore deposits comparatively large, but not suitable blast-furnace coking coals, and so the cost of coke makes ordinary blast-furnace smelting too expensive. However, it is, of course, necessary to supply carbon in some form as a reducing agent, and even though only one-third as much carbon is required for this purpose as is required in ordinary blast-furnace practice, it may be impossible to obtain even this amount at such a cost as will permit the use of the electric furnace. So far only coke and charcoal have been tried to any extent as reducing agents in the reduction of iron ores in the electric furnace, and for a time it was thought that coke could not be successfully used, as sev-

eral trial runs at Tröllhattan on coke were not successful and later the plant at Hardanger, Norway, where coke was used as a reducing agent, after being in operation for about nine months, was forced to close down, owing to the fact that with coke the quantity of pig iron that could be produced per kilowatt-year of electrical energy expended was not nearly as great as can be produced when charcoal is used. It was stated that the reason for this was that coke, especially hot coke, is a much better conductor of electricity than is charcoal. For this reason when coke is used the resistance of the charge becomes lowered, and as the smelting in the electric iron reduction furnace is done by the heat produced by the resistance encountered by the electric current in passing through the charge between the electrodes, more electrical energy is required to produce the same amount of heat. However, Profs. H. L. Vogt and P. Farup,^a of the Norwegian electro-metallurgical commission, reporting on the causes of the economic failure of this plant do not attribute the same to the use of coke, but "partly to the nature of the ores treated and partly to their low iron content, conjointly with the unsatisfactory condition of the agglomerating roasting oven, and several parts of the electrical equipment." They state that an "excellent quality of pig iron can be produced" with coke, and that "the consumption of carbon is the same whether charcoal or coke is employed;" also that the consumption of electrical energy was that originally computed by them for both poor and rich ores.

On the other hand, Engineer Gustaf Oedquist, in a statement made before the Christiania Polyteknik Forening, says:

The committee is of the opinion that the type of smelter used did not conduce to the unfavorable results at Hardanger. I am of a different opinion. The reason why it is not adapted to coke consists in the greater variation in the electrical resistance in the furnace itself by the use of coke, owing to which the average load as well as the production is diminished. To which must be added that the larger quantity of limestone thereby necessitated contributes to a lesser production of pig iron per horsepower-year.

Considerable experimenting has also been done with coke as a reducing agent by the Noble Electric Steel Co. at Heroult, Cal. In his summary of deductions Mr. Crawford, the plant manager, states:^b

1. Any ordinary grade of coke, by crushing it, can be used satisfactorily from both a metallurgical and an electrical standpoint to produce any normal grade of pig iron. But, depending on its porosity and crushing strength and its analysis, there is for each variety a size to be determined by experiment which will give from all standpoints best operating conditions.

2. Soft, porous cokes are to be preferred over hard, dense cokes, because the former can be fed in larger pieces and still insure homogeneity of mixture when the burden descends to the smelting zone. Further, they cost less to crush, and there is less loss in fines and flue dust because of the larger size which can be used. For ore crushed

^a Anon., Electrothermic iron-ore smelting in Norway: Eng. and Min. Jour., vol. 98, July 25, 1914, p. 158.

^b Crawford, John, Discussion of paper by Robert M. Keeney on "Pig steel from ore in the electric furnace.": Bull. Am. Inst. Min. Eng., June, 1914, p. 1289.

to pass a $2\frac{3}{4}$ -inch grizzly, a soft, porous coke gives best results when crushed to pass a $1\frac{1}{2}$ -inch grizzly. Certain hard, dense cokes had to be crushed to pass a $\frac{3}{4}$ -inch grizzly.

3. If the coke is of the proper size when it reaches the smelting zone, operations may be carried on with as high voltage, low current densities, good power factor, and as cool a roof when using coke as when using charcoal.

4. The power consumption per unit of production is higher with coke as a reducing agent than with charcoal, but the increase is due to the extra amount of slag made. This depends on the composition and amount of ash in the coke and the amount of sulphur in the coke.

5. The carbon consumption per unit of production is higher with coke than with charcoal. This is due partly to the greater flue-dust losses from crushed coke than from lump charcoal, and partly to the formation of carbides, in the high-lime, desulphurizing slag, all of which are not subsequently decomposed.

6. If electrodes are regulated to form a free-burning arc, the use of coke has no bearing on the electrode consumption. If, however, electrodes are inserted in the charge to form a submerged arc, as is our practice, the consumption of electrode material is considerably greater, especially with graphite electrodes. This is due principally to the greater abrasive action of coke than charcoal, though partly also to the attack of the lime slags on the electrode material. We found the electrode loss from breakage (using 12-inch graphite electrodes) very excessive, doubtless due in part to the greater density of the burden when using crushed coke as against lump charcoal. On this account, after carrying our experiments far enough to demonstrate the commercial feasibility of operating on coke, we have shut our furnaces down pending alterations to permit the use of 24-inch carbon electrodes, which, besides being stronger, will better resist the abrasive action of the burden.

7. With a coke running 1.25 per cent sulphur we had little difficulty in keeping the sulphur in the pig below 0.05 per cent, and much of the iron made was under 0.03 per cent sulphur.

8. No trouble was found in making foundry irons up to 3.50 per cent silicon or higher if desired. In fact, on account of its high, easily reduced, siliceous ash, coke will reduce a high-silicon iron somewhat more readily than charcoal.

9. Except for a somewhat more open grain, no difference has been noted thus far in the physical characteristics of iron made with coke and iron made with charcoal.

10. Crushed coke, especially when high in ash and with a large percentage of fines present, has a tendency to cause the burden to hang. This we believe would prohibit its use successfully in a furnace of the shaft type. However, in the rectangular furnaces there is an opportunity to dislodge scaffolds by barring down, if the hang is of any duration. There are several ways in which the presence of a hang-up can be detected before it has become serious. A mixture of 15 to 25 per cent of lump charcoal with the coke is quite effective in keeping the burden from hanging up.

In conclusion, we may say that coke has nothing to recommend it over charcoal for an electric-furnace reducing material, excepting the difference in price. However, "black-butts," screenings, and even breeze if the dust is removed, can be mixed to make a satisfactory reducing material. As such grades of coke can usually be secured at a very attractive price, this may more than offset the disadvantages of its use.

The writer apologizes for offering the above statements as "glittering generalities" unsubstantiated by definite figures, but, since the data obtained varied considerably with the chemical and physical properties of each coke tried and the size to which it was crushed, even a synopsis of this rather voluminous data is considered beyond the scope of this discussion.

Inasmuch as charcoal and coke are the only practical reducing agents known at present for use on a large scale, and as it would seem from what has just been stated that it is impractical to use coke in a

furnace of the shaft type, the electric iron reduction shaft furnace is necessarily limited to the use of charcoal. Hence it is necessary that such a furnace be located in close proximity to a well-timbered region, where charcoal as well as electric power can be produced at a low cost.

POSSIBLE USE OF CRUDE OIL AS A REDUCING AGENT.

In southern California and in other Western States there are rather large iron deposits of high-grade ore but coke is too expensive to permit ordinary blast-furnace smelting and the use of charcoal for electric-furnace work is entirely out of the question, as there are no forests to furnish the wood necessary for its production. On the other hand, crude oil is generally rather plentiful and comparatively cheap in such districts, especially in California.

Aside from the possible use of oil in connection with electric-furnace work, those interested in the subject have for years been considering the possibility of using crude oil as a reducing agent. So far as the authors know, the oil has not as yet been so used successfully. That the carbon and the hydrogen of the oil will reduce iron oxides is self-evident, but as yet no one seems to have been able to solve the problem of bringing the ore and the oil together at the proper temperature. As a result of a preliminary investigation conducted by the Bureau of Mines as to the possibility of using crude oil as a reducing agent, it seems as if about the only manner in which crude oil may be used for this purpose is first to convert it into a fixed gas and then to introduce this into the crucible of an electric furnace, thus preheating the gas to such a temperature that it will effectively reduce the iron oxides as it passes up through the shaft of the furnace. Be that as it may, it is to be sincerely hoped that ultimately a suitable process may be devised whereby oil may be used as a reducing agent, and thus broaden the field for the possible application of the electric furnace in the reduction of iron ores.

USE OF ELECTRIC IRON REDUCTION FURNACE AT PRESENT.

That the electric furnace has been successful in the smelting of iron ores in districts where the conditions are favorable is shown by the fact that 10 furnaces of the Swedish type with a total capacity of about 22,000 kilowatts have been erected in Sweden, Norway, and Switzerland. In this country there is one iron plant of two electric furnaces with a total capacity of 5,000 kilowatts in operation at Heroult, Cal. The type of furnace used in California is different from that used in Sweden, owing to the fact that a different grade of iron is desired from that produced in Sweden; that is, in Sweden a metal is produced that is low in silicon and carbon and particularly well suited to steel making, whereas in California the demand is for a soft gray foundry iron.

As to the advantages of the metal produced in a Swedish type of furnace for steel making, they have been stated by Keeney^a to be as follows:

1. That it can be converted into steel with greater facility and at a lower cost.
2. That it admits of greater economy in power consumption.
3. That it admits of greater economy in reducing material.
4. That the carbon content can be controlled within a limit of 2.20 per cent maximum.
5. That the losses of iron in the slag will not exceed reasonable limits, say 6 per cent FeO.

A metal has been made by the Noble Electric Steel Co., "quite similar in both appearance and analysis to the pig iron which was at that time reported as being used successfully for steel making in Sweden," concerning the use of which Crawford^b reports as follows:

Stimulated by the results which were being obtained in Sweden we made considerable effort to induce steel makers on the Pacific coast to try it. Finally by making a very attractive price we induced one firm to use it. This concern operates both basic and acid open-hearths for making concrete reinforcing steel. The report of the works superintendent on it was rather indefinite but not enthusiastic. He said "It worked pretty well, but normal pig iron was more satisfactory." Since that time we have sold this concern about 1,000 tons of normal low-silicon pig iron and also a few odd carloads of white iron which approached in analysis pig steel. They advise that no difference is noted in the working of normal electric-furnace pig iron as compared to normal blast-furnace iron of similar analysis. The pig iron brought the market price, but the pig steel always had to be offered on more favorable terms, and even at a reduced price it did not call forth any repeat orders.

These remarks are not intended to be construed as a contradiction of the favorable reception which pig steel has been accorded by the Swedish steel men, but it does seem to indicate that to gain the advantages claimed pig steel must be used "according to prescription" and a good deal of "missionary work" might be necessary before American steel makers, who have no patriotic interest in the matter, would do the necessary experimenting to prove the advantages claimed.

ENGINEERS' REPORT OF THE EXPERIMENTAL WORK AT TROLLHÄTTAN.

The engineers in charge of the experimental work at Trollhättan made the following statements in their final report according to the Iron and Coal Trades Review:^c

QUALITY OF PIG IRON PRODUCED IN ELECTRIC FURNACE.

The silicon content does not vary more than in an ordinary blast furnace. The phosphorus content is lower than with the same quality of charge in an ordinary blast furnace; this is due to the lower consumption of charcoal. The percentage of sulphur is, however, slightly higher in the electric furnace. But it should be observed that both at Trollhättan and at Hagfors unroasted ores have been used without any difficulty arising from the sulphur present.

^a Keeney, R. M., Pig steel from ore in the electric furnace: Bull. Am. Inst. Min. Eng., February, 1914, pp. 349-367. Discussion, Ibid, J. W. Richards and J. Crawford, p. 1289.

^b Crawford, J., Discussion of paper by R. M. Keeney on "Pig steel from ore in the electric furnace:" Bull. Am. Inst. Min. Eng., June, 1914, p. 1289.

^c Iron and Coal Trades Review, 1913.

The quality of the electric pig iron has been highly commended. It acts particularly well in the open-hearth furnace, and steel made from it is certainly not inferior to steel made from ordinary pig iron. Mr. E. Odelberg, managing director of the Stromsnas Iron Works, states that the electric pig iron is "of the very best quality for the open-hearth process, and, as regards the uniformity of the silicon content, fully as uniform as iron from an ordinary blast furnace." Mr. A. Herlienus, managing director of the Uddeholm Co., states that "the electric pig iron has been used with satisfactory results, both for the open-hearth, Bessemer, and the Lancashire processes. Generally speaking, there has been no difficulty in obtaining pig iron of uniform quality, although slightly better uniformity may possibly be obtained with a very carefully conducted blast furnace."

It has been found that the proportion of concentrates ought not to exceed 20 per cent of the ore charged. This figure, however, does not appear to be final, as in a later and somewhat modified furnace of the same type at the Hagfors Iron Works 25 per cent of concentrates is used without any difficulty.

The power consumption per ton of pig iron varies in proportion to the iron content in the ore. A poor ore or pig iron high in silicon and manganese requires more power than a rich ore or a pig iron low in silicon and manganese. For such iron the power consumption averages only 2,067 kilowatt-hours per ton of pig iron; that is, there are obtained 4.22 tons of pig iron per kilowatt-year, or 3.10 tons per horsepower-year.

The charcoal consumption per ton of pig iron varies from 20 to 24 hectoliters (56 to 68 bushels), depending on the quality of the charcoal and the charge. Coke has been found to be unsuitable for this furnace unless mixed with charcoal.

The consumption of electrodes at Trollhättan has been reduced to less than 3 kilograms (6.6 pounds) per ton of pig iron. At Hagfors it has amounted to as much as 6 to 9 kilograms (13.2 to 19.8 pounds). This discrepancy is explained by the fact that the electrode consumption is increased in proportion to the higher power consumption for a poorer charge, and is further increased by the more efficient circulation of gases and higher CO_2 content in the gas. The lower electric load per unit of surface at the Hagfors furnaces also contributes to the higher electrode consumption at this plant.

The cost of repairs is lower than there appeared at first to be reason to expect. In the manufacture of pig iron containing silicon and manganese the costs for repairs are higher than in producing pig iron with low contents of those elements.

VALUE OF GAS PRODUCED.

At the plant of the Uddeholm Co., at Hagfors, the gas from the furnaces has been used with good results for heating the open-hearth furnaces. It has been estimated that the value of the gas obtained per ton of pig iron may be taken at 2.50 kroner (about 0.42 cent). We quote from the report a summary of the most important figures relating to the economical results. These show that step by step the results have been improved and the quantity of iron per horsepower-year increased, whereas the electrode consumption and time for repairs have been reduced, these three items in conjunction with the saving in charcoal being the decisive factors as regards electric iron smelting.

The figures show that during the last few months as much as 3.20 to 3.10 tons of iron has been obtained per horsepower-year, whereas before the alterations then made the highest average figure was 2.86 tons. The highest expected yield was 3 tons per horsepower-year, a figure that has thus been exceeded. It may also be mentioned that during single periods of several weeks while especially suitable ores were used the highest average figures above mentioned have been materially exceeded. It will thus be seen that as regards efficiency the results of the operation of the furnace have surpassed expectations. The electrode consumption was 13.8 kilograms (30.4 pounds), which has finally been reduced to about 3 kilograms (6.6 pounds) per ton of pig iron. The cost and time required for repairs could not be estimated beforehand.

Experience has shown that both the cost and the time required are less than could have been hoped for. Wherever employment for the gas can be found this should also be taken into account, as is being done at Hagfors, where its value for firing the open-hearth furnaces is estimated to reduce the cost of the pig iron by about 66 cents.

The results of the work done at Trollhättan in reducing iron ores in the electric furnace are summarized in the following table:

Summary of results obtained at Trollhättan.

Item.	Nov. 15, 1910, to May 29, 1911.	Aug. 4, 1911, to June 21, 1912.	Aug. 12 to Sept. 30, 1912.	October to Decem- ber, 1912.
Ore, concentrates, and briquets, kilograms.....	4, 338, 338	7, 917, 214	1, 406, 530	2, 914, 830
Limestone, kilograms.....	345, 405	647, 479	108, 150	169, 944
Charcoal, hectoliters.....	65, 474. 5	107, 282. 5	21, 859. 5	44, 934. 5
Coke, kilograms.....		70, 854		
Electric energy, kilowatt-hours.....	6, 339, 131	10, 845, 180	1, 939, 073	3, 957, 565
Iron content of ore, per cent.....	60. 79	60. 75	68. 67	65. 38
Iron produced, kilograms.....	2, 636, 098	4, 809, 670	965, 915	1, 005, 865
Slag per ton of iron, kilograms.....	350	324	192	
Electrodes per ton of iron, gross, kilograms.....	10. 00	6. 08	3. 02	2. 78
Electrodes per ton of iron, net, kilograms.....	4. 95	5. 17	3. 02	2. 78
Charcoal per ton of iron, hectoliters.....	24. 84	22. 31	22. 63	23. 58
Working time, hours and minutes.....	4, 441 20	7, 218 23	1, 173 8	2, 158 30
Time used in making repairs, hours and minutes.....	236 53	506 7	13 47	49 30
Time used in making repairs, per cent of total time.....	5. 06	6. 55	1. 16	2. 24
Average load, kilowatts.....	1, 427	1, 502	1, 653	1, 833
Time per ton of iron, kilowatt-hours.....	2. 405	2. 255	2. 007	2. 076
Iron per kilowatt-year, tons.....	3. 64	3. 88	4. 36	4. 22
Iron per horsepower-year, tons.....	2. 68	2. 86	3. 20	3. 10

COPPER.

PRESENT STATUS OF THE ELECTRIC SMELTING OF COPPER ORES.

So far as the writers know, the smelting of copper ores in the electric furnace is not done commercially in the United States at the present time. It is reported, however, that in Norway smelting of copper ores with an electric furnace of 750 kilowatts and an estimated capacity of 2,000 tons of ore per annum has been tried at the Ilen Smelting Works, Trondhjem, and it is understood that copper ores are to be smelted regularly at this plant. A consignment of 25 tons of pure copper, the first copper ever produced by electric smelting and refining in Norway, was recently exported from the Bitovaria mine at Kaafjord, near Dyngor. Some months ago it was reported that the electric smelting and refining of nickel and copper would be undertaken by a company at Christiansand.

EXPERIMENTAL WORK OF VATTIER, SCHILOWSKI, WOLKOFF, AND OTHERS.

In addition to the work on a commercial scale attempted at Norwegian plants, experimental work on the electric smelting of copper ores has been done by Vattier, Schilowski, Wolkoff, Stephan, and the writers. Vattier ^a smelted an ore containing 7 per cent copper and

^a Haanel, E., Report of the commission appointed to investigate the different electrothermic processes for the smelting of iron ores and the making of steel in operation in Europe: Canada, Department of Interior, Mines Branch, 1904, p. 215.

8 per cent sulphur with the production of a matte containing 47.9 per cent copper. Schilowski^a smelted mixtures of raw ore and roasted ore in an electric furnace and produced a matte. He found that the consumption of energy was lowest when the charge contained 60 to 70 per cent of roasted ore. Using an ore containing considerable arsenic, he found it possible to condense the volatilized arsenic in chambers. Wolkoff^b treated a mixture of copper sulphide (Cu_2S) and copper oxide (CuO) in the electric furnace for the production of copper by the reaction of the sulphide and the oxide. Stephan^c smelted oxidized copper ores in the electric furnace for the production of metallic copper with charcoal as a reducing agent. In their experimental work the writers have smelted native copper concentrates and sulphide ores of copper in the electric furnace with favorable results.

In view of the results of the experimental work of the writers and others, there seems no good reason why copper-bearing ores can not be as successfully treated in an electric furnace as in a combustion furnace. In all combustion furnaces used for the treatment of copper ores, the fuel takes no part in the reactions necessary for the production of the desired product, unless in the reduction of oxide ores smelted alone, that is, without a mixture of sulphides, a step that is practically unknown in this country at present. For example, in the reverberatory furnace the fuel acts only as the heating agent; in ordinary blast-furnace smelting, the coke is used to supply the heat necessary to raise the temperature so as to permit the necessary reactions between the oxides, sulphates, and sulphides of the charge, and to scorify the resultant mass, separation of the slag and the matte being thus permitted. In partial pyritic and in pyritic smelting the necessary oxidation of the sulphides and iron is produced by the oxygen of the air entering at the tuyères, and the coke used is simply for supplying the amount of heat necessary for carrying out the process, which is not supplied by the oxidation of the sulphur and the iron present in the charge at the time it passes through the tuyère zone of the furnace. Such being the case, there seems to be no reason why the smelting of copper ores could not be done just as well by electric heat as by that derived from the combustion of coke, especially if warranted by local conditions.

SUMMARY.

Briefly, then, the electric smelting of copper ores is nothing more than the substitution of electric heat for the heat derived from the

^a Schilowski, J., *La fusion électrique de cuivre et des produits intermédiaires de fonderies de cuivre*: *Revue de Métallurgie*, vol. 9, 1912, p. 205.

^b Wolkoff, W., *Electric smelting of copper sulphide ore*: *Metallurgie*, vol. 7, 1910, p. 99.

^c Stephan, M., *Einiges über die Erzeugung von Metallen im elektrischen Ofen*: *Metall und Erz*, vol. 1, 1913, p. 11; *Met. and Chem. Eng.*, vol. 2, 1913, p. 22.

combustion of carbon as produced in blast-furnace smelting. In some cases the reactions of copper smelting would take place to better advantage in the neutral atmosphere of the electric furnace than in the reducing or partly reducing atmosphere of the combustion furnace. Therefore as to whether the electric furnace should be used for the smelting of copper ores will largely depend on the relative cost of coke and electric power. As the use of the electric furnace is not advocated as a competitor of the combustion furnace, but as a substitute for it in those localities where it is not advisable because of the high cost of fuel to use a combustion furnace, there is no apparent reason why the electric furnace may not be developed as a substitute, especially in the treatment of copper-bearing ores. In this connection it is to be remembered that the reason why the electric furnace was developed in the iron industry for the reduction of iron from its ores was one of necessity. As a matter of fact the field for the electric furnace in the reduction of iron from its ores is limited. Perhaps the same is true as regards the possible application of the electric furnace to the treatment of copper ores, but, judging from the comparative costs, it seems that the chances in favor of the electric furnace for the treatment of copper ores are greater than those for its use in the treatment of iron ores, because there is not so great a difference in the cost of coke and electric power in the copper-mining districts as in the iron-smelting centers. Also the cost of electric power is constantly becoming less, owing to improvements in gas engines and steam turbines, so that in districts where water power is not plentiful, but cheap fuels unsuitable for coke making are available, it may be more advantageous to use electric heat than the heat derived from the combustion of coke.

LEAD ORES AND COMPLEX SULPHIDE ORES.

The smelting of straight lead ores in the electric furnace has seemingly never been attempted either commercially or on a large experimental scale, largely because of the ease and cheapness of smelting such ores by combustion processes. For the smelting of ordinary lead ores it has no especial application, but in the treatment of complex sulphide ores the electric furnace might be profitably used.

As stated elsewhere in this report, ores of lead, zinc, and silver are being electrically smelted at Trolhättan. During the smelting the lead collects in the bottom of the furnace and acts as a collecting agent for the silver; the zinc is volatilized and is subsequently condensed. In mining districts of the Western States are complex ores of lead, zinc, copper, gold, and silver that might be profitably treated in this manner, except that if copper were present the process would be further complicated by the formation of a matte. With the solution

of the zinc-condensation problem in electric-furnace work, it is not unreasonable to suppose that such complex ores can be electrically smelted with a fair recovery of the base metals, and with a high recovery of the precious metals. Also, owing to the fact that practically as high a temperature as may be desired may be obtained in the electric furnace, it would be possible to smelt complex ores that are high in gold and silver content but low in base metals, such as lead and zinc. Such recovery would be worth the additional expense of wet concentration. In the blast furnace 10 per cent of zinc in the charge causes sticky slags, and a much higher percentage makes the operation of the furnace irregular and difficult. Such slags can be fused readily in an electric furnace. If the ore contained copper and zinc, the ore could be charged into an electric blast furnace, matte being used as an agent for collecting the gold and silver.

GOLD AND SILVER ORES.

In the smelting of gold and silver ores carrying no lead or copper to form either a lead bullion or a copper matte, but containing iron sulphide, the electric furnace might be used in connection with an air blast for oxidation of the iron sulphide, an iron matte being used as a collecting agent for the gold and silver. The value of iron matte as a collecting agent is still largely a matter of personal opinion, but from the pyritic smelting done with iron as a collecting agent and from some of the writers' experiments in which iron matte was used in the electric furnace, it is evident that a fair recovery can be made in this way. As in electric copper smelting, the electric-furnace treatment of these ores is still in the experimental stage.

Recently a plant has been installed at Lluvia de Oro, Chihuahua, Mexico, for the electric melting of zinc cyanide precipitate and the smelting of ore concentrates.^a The gold-silver bullion resulting is shipped and the slag resmelted or concentrated. The process appears to be successful in this application.

The writers believe that at many mines in isolated districts where fuel is usually high and where hydroelectric power is used for the mine or mill, the cyanide precipitates could be melted more economically in electric furnaces than in combustion furnaces. The construction of a small furnace of the Siemens type is simple and inexpensive. By using the electric furnace when one of the units of the mill is idle for repairs or during the mine shift when little hoisting or drilling is being done, no additional electrical installation would be necessary. Thus, power that under ordinary conditions is wasted will be utilized.

^a Coulsin, H. R., Electric furnace at Lluvia de Oro: Eng. and Min. Jour., vol. 93, 1912, p. 1189.

ZINC.

In the metallurgy of nonferrous metals the electric furnace has had a greater application for the treatment of zinc ores than in the metallurgy of any of the other nonferrous metals except aluminum. Since 1885, when an electric furnace for the treatment of zinc ores was patented by the Cowles brothers, experimental work has been done. However, the process has not been applied to any great extent because of the difficulty of condensing the zinc vapor produced in smelting in the electric furnace. The cause of this difficulty has not yet been definitely determined. With few exceptions the work has not been done on a very large scale, and so it may be said that the electric smelting of zinc ores is still in the experimental stage.

Undoubtedly there is an inviting field for the electric furnace in the metallurgy of zinc, but before its wide application can be expected the condensation problem and various mechanical problems must be solved. The latter should not prove difficult. The special field for electric-furnace work in zinc metallurgy is due largely to the low thermal efficiency of the present zinc retort, which is given by Richards ^a as only 7 per cent.

As the efficiency of the electric furnace is 50 to 75 per cent, the difference in the heat necessary for carrying out the smelting operation is so great as to permit a higher cost of electrical energy than is permissible in many electrometallurgical operations.

In this report the writers discuss broadly and briefly the present methods of retort smelting, experimental work on the electric smelting of zinc ores, and the present status of electric zinc smelting. The Bureau of Mines is investigating the electrometallurgy of zinc ores, and the results of experimental work, together with a complete résumé of literature on the subject, will be presented at a later date.

PRESENT METHODS OF ZINC SMELTING.

Briefly, the present method of zinc smelting consists of the roasting of sphalerite (ZnS) to zinc oxide (ZnO). The roasted ore is then smelted with excess carbon as a reducing agent in a closed fire-clay retort having a condenser attached. The zinc oxide is reduced to zinc at a temperature of $1,033^{\circ}\text{C}$. and as the boiling point of zinc, according to various authorities, is from 920° to 930°C ., the reduced zinc passes off as a vapor and is condensed as liquid metal in a condenser. In order to make the reaction more rapid the temperature of the retort is kept at $1,100^{\circ}$ to $1,300^{\circ}\text{C}$.

Zinc vapor condenses into a liquid mass in the range of temperature from 420° to 860°C . The lower temperature is that of solidification of zinc, and the upper approaches the vaporization point. In practice

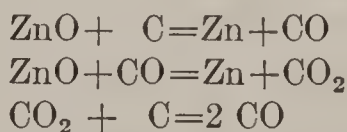
^a Richards, J. W., Metallurgical calculations, pt. 1, 1906, p. 80.

it is the aim to keep the temperature at 500° C., so that the zinc can be tapped off molten from the retort. If the temperature of the condenser is too low the vapor is too dilute, or if the vapor is diluted with some carbon dioxide instead of carbon monoxide, or with metallic vapors other than zinc, there will be a formation of blue powder, which consists of minute particles of zinc, each particle seemingly covered by a thin coating of zinc oxide. Because of this coating of oxide the powder can not be converted to spelter by simple melting.

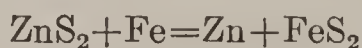
The process is intermittent and a retort holds only about 200 pounds of mixture. A charge is smelted, the zinc removed from the condenser, and the residue raked out of the retort. Because of the small scale and intermittent nature of the operation the labor cost is high. Likewise the thermal efficiency is low because of the heat being conducted to the charge through the furnace walls. The retorts are an expensive feature of the process, as at the high temperature necessary they last only about a month.

ZINC SMELTING IN THE ELECTRIC FURNACE.

Processes for the electric smelting of zinc ores are of two general classes—first, reduction of the zinc oxide and its compounds by carbon and carbon monoxide, and, second, decomposition of the sulphide by metallic iron. Examples of the first method are the De Laval and Johnson processes, and of the latter, the Cote-Pierron and Imbert-Thomson-Fitzgerald processes. Reduction of zinc oxide with carbon is based upon the reactions



The use of iron is according to the reaction



In neither of these processes does the electric current perform any function other than heating. As alternating current is supplied to the furnace, there is no electrolysis. As is shown later, the process may be performed in either an arc or a resistance furnace.

One essential difference between the reduction of zinc oxide with carbon in an electric furnace and in the retort is that the retort process is intermittent, whereas the electric process may be continuous. In the electric furnace the charge is added at intervals, as may be necessary, and the slag is tapped or allowed to run continuously from the furnace without disturbing the operation of the process. The reduction by carbon or carbon monoxide occurs as in the retort, except that owing to the rapidity of reduction in the electric furnaces as at present designed the third reaction seemingly does not take place as well in the electric furnace as in the retort, so that the zinc vapor con-

tains more carbon dioxide, which has a bad effect on the condensation and results in the production of a large amount of blue powder. Various attempts have been made to obviate this evil, the principal one being the passing of the vapor through incandescent carbon before it enters the condenser.

ELECTRIC ZINC SMELTING AT TROLLHÄTTAN AND SARPSBORG.

The largest plant in commercial operation for the electric smelting of zinc ores is that of the Norse Power & Smelting Syndicate at Trollhättan, Sweden, which also has a smaller works at Sarpsborg, Norway.

EQUIPMENT AT THE TWO PLANTS.

The Trollhättan plant has a furnace house 300 by 52 feet, which contains 11 resistance furnaces, 6 on one side of the building and 5 on the other. There are also two refining furnaces and several arc furnaces. In 1912 it was reported that the new furnace house, then partly erected, was to contain eight furnaces of 600 to 900 kilowatts each. The old furnaces each have a capacity of 270 kilowatts. The total power now being used at the works is 3,000 kilowatts, and with the new installation it will be 7,500 to 10,000 kilowatts.

The Sarpsborg plant has three arc furnaces and four refining furnaces.

For the past six or seven years zinc smelting has been conducted at these works in the De Laval furnace, but part of the product has been the result of refining dross and scrap. It is stated that now ore smelting, in which carbon is used as a reducing agent, is being carried on and that regular shipments of finished zinc were made during the year 1912. The only information available about the works is contained in Harbord's report ^a published in 1911, as everything in regard to the process and the results obtained is kept as secret as possible.

SMELTING PROCESS.

The smelting process consists essentially of two operations, concentration of the zinc to a rich oxide, followed by reduction in another furnace. Oxide ore, flux, and carbon used as reducing materials, are charged into the furnace, where most of the zinc and some of the lead are volatilized and condensed, partly as metal and partly as blue powder and oxide, containing 54 per cent zinc and 20 per cent lead. This is really a concentration process, for the blue powder is subsequently mixed with ore and recharged, when a larger percentage of the metal volatilized is recovered as metal. The remainder of the lead, which carries some silver, is reduced to metal in the smelting hearth and is tapped with the slag. Some lead, zinc, and silver passes into the matte and some into the slag.

^a Harbord, F. W., Zinc smelting at Trollhättan: Eng. and Min. Jour., vol. 93., 1912, p. 344.

Each furnace is of a modified Siemens type with a shallow hearth and covered with a roof. A carbon block bottom forms one electrode and the other passes vertically through the roof. In some furnaces the ore is charged through the roof, in others at one side of the electrode, whereas others have been designed having a continuous side feed. Three-phase current is transformed from 10,000 volts to 110 volts at the furnaces. The furnaces are connected one to a phase. The capacity of a 270-kilowatt resistance furnace is 2.8 tons of ore per 24 hours. Each furnace holds about 3 tons. A condenser is attached to each furnace.

The charge used by Harbord consisted of 600 pounds of roasted Broken Hill slime, 22 pounds of calamine, and 165 pounds of coke. Broken Hill slime averages 32 per cent zinc, 23 per cent lead, and 25 ounces per ton silver. The powder resulting from this charge was resmelted in another furnace, the charge being 220 pounds of roasted Broken Hill slime, 440 pounds of powder, 55 pounds of coke dust, and 11 pounds of lime. Three furnaces were operated on ore mixture, and four furnaces on ore-powder mixture. At the end of the second day the ore furnaces were making sufficient powder to supply the powder furnaces. The furnace condensers were tapped every four hours for crude zinc and powder, and the slag, matte, and lead were tapped about every eight hours.

RESULTS OF OPERATION OF FURNACES.

During the first part of the run the slags were not clean because of imperfect separation of matte and slag. This fault was caused by the use of too basic a slag and by imperfect mixing of the charge. Toward the end of the work this was improved considerably. There was also considerable trouble experienced in changing electrodes, owing to defective mechanical apparatus, a difficulty that could easily be remedied in a new plant. There were no serious stoppages caused by failure of the furnaces.

In 27.58 days the seven furnaces smelted 578 metric tons of roasted Broken Hill slimes, 19 tons of calamine, and 22.5 tons of blue powder from stock. From this material there was produced 160.8 tons of crude zinc and 36 tons of powder. At the end of the run there was 13.4 tons more powder on hand than at the beginning. The crude metal averaged 79 per cent zinc, 20 per cent lead, and 0.6 per cent iron. From this there was produced on refining 112.4 tons of spelter containing 99.9 per cent zinc and 24.7 tons of lead. The lead tapped with the slag was remelted to remove the slag. This yielded 41 tons of marketable bullion, containing 141 ounces of silver per ton. There was also obtained 17 tons of leak lead, assaying 27 ounces silver per ton. In addition there was 9 tons of skimmings which contained lead, zinc, and silver. The total input of metals was 204.04 tons of

zinc, 128.35 tons of lead, and 15,750 ounces of silver. Thus with 537 tons of ore charged the charge averaged 38.1 per cent zinc and 24 per cent lead. The extraction as metals was 130.46 tons of zinc, 94.94 tons of lead, and 7,230 ounces of silver. The yield as metals was then 64 per cent zinc, 73.99 per cent lead, and 45.9 per cent silver. Including the metals in the powder left the yield was 73.4 per cent zinc, 79.3 per cent lead, and 49.5 per cent silver.

At the Sarpsborg plant the electrothermic-refining results on the crude spelter from the powder furnaces were as follows: Fine zinc 83.9 per cent, powder 9.8 per cent (corresponds to 8.8 per cent metals), residues of lead, etc., 2.2 per cent, charge in furnace 1.1 per cent, impregnations in brick walls of furnace and condenser 0.5 per cent, unaccounted for 3.5 per cent, total 100 per cent. The effect of a new lining is shown by the loss unaccounted for, being 5.7 per cent at Trollhättan during the experimental run, as compared with 3.5 per cent in regular running at Sarpsborg.

The consumption of electric current in the experimental run at Trollhättan averaged 2,078 kilowatt-hours per ton of ore smelted, which does not include current used in resmelting the powder. Two tons of powder was retreated per ton of ore smelted. A comparison of the resistance furnaces at Trollhättan with the arc furnaces at Sarpsborg shows that while one was as good as the other for producing the desired reaction, the energy consumption was 70 per cent more in the arc than in the resistance furnace. The consumption of electrodes was also higher in the arc than in the resistance furnace, being 89.2 pounds per ton of ore at Sarpsborg and 69.3 pounds per ton at Trollhättan.

COMMENTS BY HARBORD AND MOULDEN.

Regarding the process, Harbord ^a reports as follows:

The weak point of the process is the large amount of metallic powder produced in proportion to the metals. The recovery of a large percentage of metals is admittedly a very difficult problem, but that present practice can be very greatly improved upon I have not the least doubt, leading to decreased consumption of energy and reduction of labor costs. One detail which is a great improvement on the early practice is the better separation of the crude zinc from the powder by a mechanical stirrer.

At the time Harbord made his report, J. C. Moulden,^b who is connected with a company (the Sulphide Corporation) that ships the Broken Hill slimes used at Trollhättan, stated that he was of the opinion that the extraction, after certain improvements should be made, would eventually be 75 per cent zinc, 80 per cent lead, and 80 per cent silver.

The cost of power at Trollhättan is about \$10 per kilowatt-year or 0.11 cent per kilowatt-hour.

^a Harbord, F. W., Zinc smelting at Trollhättan: Eng. and Min. Jour., vol. 93, 1912, p. 344.

^b Met. and Chem. Eng., vol. 9, 1911, p. 673.

EXPERIMENTS AT M'GILL UNIVERSITY.

For several years extensive experimental work on a laboratory scale has been conducted under the direction of Dr. Alfred Stansfield and Mr. W. R. Ingalls.^a This experimental work has been largely directed toward a study of the condensation problem and the speed of reduction. Uncommonly high percentages of carbon dioxide were found in the gases from the electric furnace, which, as previously stated, prevents condensation of the zinc as metal by oxidizing the condensing globules of the zinc and preventing coagulation. Other experimenters have noted this difficulty, as well as the presence of considerable quantities of carbon and ore dust in the condenser. The latter are believed to be due to the stirring up of dust in feeding the charge into the furnace.

Experiments were also conducted on the reduction of zinc oxide in an intermittently heated retort. The charge was introduced into the retort at the beginning of the operation and at the end the residue was discharged as an unfused residue. Reduction took place slowly and the results obtained showed that although it might have been possible to operate the process continuously, such a course was not feasible. When the furnace was started with a bath of slag and the charge introduced continuously upon the bath, the reduction became relatively rapid. No explanations were given for this increase, but Ingalls states that a retort of small proportions smelts a surprisingly large quantity of charge by comparison with the retort of ordinary practice at an equivalent temperature. With a furnace consisting of a plumbago crucible 12 inches in diameter, 10 pounds of ore per hour was smelted. With a furnace 18 by 18 inches in horizontal section, 737 pounds was smelted in 23 hours. However, the statements as published are not complete. The output of an electric furnace is due chiefly to the energy input and not to the interior volume of the furnace. The capacity in kilowatts of the furnaces used is not given, but it is clear that, in comparison with a combustion retort, the electric furnace has a high output per unit of interior volume.

As the electric furnace may have a high temperature, most of the zinc may be driven off, a low zinc slag being left to be tapped from the furnace. A calcareous slag as well as a high temperature is also necessary. In the experiments at McGill University, the slags contained less than 1 per cent of zinc.

Interesting figures of power consumption were obtained in the experiments at McGill. In one run lasting 24 hours, with a resistance furnace 18 by 18 inches, the power consumption was 1,610 kilowatt-hours per short ton of ore smelted. Theoretically it is estimated that 955 to 1,060 kilowatt-hours per short ton of ore is necessary to

^a Ingalls, W. R., The electric smelting of zinc ores: Eng. and Min. Jour., vol. 94, 1912, p. 7.

smelt ordinary zinc ore. Ingalls concluded that a figure of 1,200 kilowatt-hours per short ton would be ample. He also concluded that there is some chance for electric zinc smelting if such smelting can be done with 1,200 kilowatt-hours per ton of ore, if the labor and electrode costs can be reduced materially, and if 90 per cent of the metals can be extracted.

Previous to the experiments of Stansfield and Ingalls, Stansfield and Reynolds conducted experiments on the smelting of lead and zinc sulphide ores carrying silver, with the intention of distilling and condensing the zinc, and reducing the lead in the molten form, thus collecting the silver. The experiments showed the possibility of smelting the ores in the manner indicated, and the extraction of lead was particularly satisfactory. The later work of Stansfield and Ingalls verified the former results and ores of this class are now being treated at Trollhättan.

EXPERIMENTS OF JOHNSON.

One of the most persistent investigators of the electric smelting of zinc ores is W. M. Johnson,^a who has conducted experiments on a large laboratory scale for several years. His furnace has vertical upper electrodes of carbon and a carbon conducting hearth. The ore is fed through the roof of the smelting chamber. There are several tap holes at different levels. The zinc vapor passes through a flue into an adjacent column of carbon which is heated by the passage of the electric current. During the downward passage of the vapor through the hot carbon, the carbon dioxide is supposed to be reduced to monoxide. The vapor then passes into a condenser, where most of the zinc is condensed as metal and some as blue powder. The furnace is operated continuously. The ore is preheated by gas to 900° C. before entering the electric furnace.

Except for a few statements, the information printed about the work done by Johnson does not clearly show the extent to which he has carried condensation without the formation of blue powder, but the use of a column of hot carbon to reduce the carbon dioxide in the vapor seems to have helped to lower the percentage of blue powder formed.

ELECTRIC SMELTING OF ZINC SULPHIDE ORES WITH IRON AS A DESULPHURIZING AGENT.

The two chief exponents of the use of iron as a desulphurizing agent in the electric smelting of zinc sulphide are the Imbert and Cote-Pierron processes. Imbert tested the reaction, without much success, in the combustion furnace. Thomson and Fitzgerald later designed an electric furnace for its application.

^a Johnson, W. M., Notes on electric zinc smelting: *Met. and Chem. Eng.*, vol. 10, 1912, p. 537.

Although the Imbert-Thomson-Fitzgerald process has been tried experimentally on a seemingly large scale for two years in Upper Silesia, little authoritative information has been issued concerning it. Several types of resistance furnace have been used, all having a resistor of solid carbon.

The Cote-Pierron process^a uses a furnace of the Siemens type, or one having the two vertical electrodes in series, both types being roofed and connected with a column of electrically heated carbon, through which the vapor passes before going to the condensing chamber.

The same trouble has been experienced in using iron as in using carbon for reduction of the zinc oxide, namely, the proportion of blue powder formed. It seems that iron and iron sulphide are volatilized and that other vapors are given off which dilute the zinc vapor and cause trouble in the condensation of the zinc. Even with the column of carbon of the Cote-Pierron process this trouble continued. Moreover, the Cote-Pierron process does not appear to have developed into the commercial success it was expected to be.

Although it is claimed that the use of iron as a desulphurizing agent in the direct treatment of sulphide ores avoids roasting, it is probable that any saving thus effected is more than offset by the cost of iron. For carrying out the reaction, about 700 pounds of iron is stated to be necessary per ton of 30 per cent zinc ore. At \$12 per ton the cost of iron would be about \$4 per ton of zinc sulphide ore treated. The cost of roasting a zinc sulphide ore is estimated by Ingalls at \$1.25 to \$1.75 per ton. It is doubtful whether a regenerative process for recovering the iron from the iron sulphide formed would be successful, or cheap enough to reduce the cost below that of roasting. The zinc sulphide might be sold if a market was near, but it probably would be contaminated by other metals in the ore.

PRESENT STATUS OF ELECTRIC ZINC SMELTING.

Although, as previously stated, more progress has been made to date in the electric smelting of zinc ores than with the electric smelting of any of the nonferrous metals except aluminum and metals forming ferro-alloys, such as silicon, chromium, and tungsten, nevertheless the process is still largely in the experimental stage. There is no plant operating on a commercial scale except the Trollhättan works, which take 7,500 to 10,000 kilowatts. The experimental work continues, but as yet none of it has developed a commercial operation.

It was recently reported that the Sulphide Corporation had constructed, at Cockle Creek, New South Wales, an electric zinc smelting

^a Fleurville, E., Electric zinc smelting: *Met. and Chem. Eng.*, vol. 7, 1909, p. 468.

plant, combined with a plant for the manufacture of sulphuric acid and superphosphate. The plant was supposed to be ready for operation in June, 1913, but no later information has been received.

DIFFICULTY OF CONDENSING ZINC VAPOR INTO METAL.

From what has been said concerning the work at Trollhättan and the results of others, it is evident that the difficulty lies almost entirely in the condensation of the zinc vapor into metal rather than with the formation of blue powder under the peculiar conditions of the electric furnace. The electric furnace, mechanically or electrically, presents no great difficulties, because all of the troubles formerly experienced have been solved in the construction of large pig-iron, steel, carbide, and ferro-alloy furnaces. The problem, then, is one of a metallurgical nature and is caused by the different conditions and greater speed of smelting in the electric furnace than in the combustion retort.

THE ELECTRODE PROBLEM.

The opponents of the electric smelting of zinc ores, who as a rule appear to have no knowledge of the great development of the electric furnace itself in the treatment of other ores, often state that although the retort is the weak part of the old system, so is the electrode of the electric method. Judging from large-scale electrometallurgical work on other ores the authors do not believe that in general the electrode is an exceptionally weak part of electric smelting. The electrode problem is no longer serious in iron smelting, and in other electric-furnace work. It is only fair to assume that the designer of an electric zinc smelter should stand on the shoulders of accumulated knowledge and utilize the results of others when the metallurgical problems of electric zinc smelting have been solved.

RETORT AND ELECTRODE CONSUMPTION.

Although it is not the purpose of the writers to enter into a discussion of the relative advantages of processes, it may be of interest to present a few comparative facts regarding retort and electrode consumption.

Ingalls^a states that during a period of ten years the average life of a Silesian retort was 38 days. One charge was melted per day. The Silesian retort is said to take 200 pounds of ore per charge.^b Hence a retort treats 7,600 pounds or 3.8 tons of ore. Ingalls^c also states that a condenser lasts 8 to 12 days, so that about four condensers are necessary for the ore smelted by a single retort. He also states that a seasoned retort is valued at 50 cents and a condenser at

^a Ingalls, W. R., *The metallurgy of zinc*, p. 545.

^b Editorial: *Eng. and Min. Jour.*, vol. 94, Dec. 14, 1912, p. 1109.

^c Ingalls, W. R., *op. cit.*, p. 242.

4 cents. Thus, the cost of a retort and condensers for smelting 3.8 tons of ore is 66 cents, or 17 cents per ton of ore smelted.

At Trollhättan, Sweden, the electrode consumption has been reduced to 6 pounds per ton of pig iron produced. Considering that 2 tons of ore was charged for each ton of pig iron produced, the consumption per ton of ore is 3 pounds. The smelting conditions in electric iron reduction would not tend to be as strongly reducing as in the zinc retort, although the ore in each case would be an oxide. Hence it may be expected that the total consumption of electrodes in a large properly designed zinc furnace will not exceed 3 pounds per ton of ore. There is no longer loss of electrode because of unconsumed ends, as that loss has been entirely overcome by jointing both amorphous-carbon electrodes and graphitized electrodes. This procedure reduces the loss by one-half. In some experiments by the writers on a mixture of sulphide and oxidized copper ores with no carbon in the charge and the charge proportioned so as to contain 15 per cent sulphur, the electrode consumption with air blown into the furnace for partial pyritic smelting was only 2 pounds per ton of ore. However, the furnace was designed so that the charge protected the electrode from the air. Amorphous-carbon electrodes can be obtained at 3 to 4 cents per pound, so that on the basis of a consumption of 3 pounds per ton of ore the electrode cost would be 9 to 12 cents, as compared with a retort cost of 17 cents. These figures seem to show that the retort at the present time is the weaker.

THE FUNDAMENTAL DIFFICULTY.

Then, if the failure of the electric furnace in smelting zinc ores is not due to the electric furnace nor the electrodes, what is the metallurgical condition that causes condensation difficulties and has retarded the commercial application of the process? The difficulty is of course due to a difference in the smelting conditions of the electric furnace as compared with the retort. Both physical and chemical conditions influence the formation of blue powder. Some powder forms under certain conditions of pressure and temperature. Just what these conditions are has not been determined. The problem is one for the physical chemist. The formation of blue powder is due largely to the speed of reduction of zinc oxide in the electric furnace and to the presence of volatile impurities in the charge. Boudouard found that the permissible percentage of carbon dioxide in the zinc retort was low, and that when the limit was exceeded the carbon oxidized the zinc, causing the formation of the blue powder. The vapor of the zinc retort is usually below this limit or not much above it, so that the zinc is condensed chiefly as liquid zinc. But in the electric furnace, the temperature directly beneath the electrode in either the arc or the resistance furnace is so high that the reduction

proceeds rapidly and the large amount of carbon dioxide formed does not have a chance to be reduced to monoxide by the hot coal of the charge before entering the condenser. Hence, there is the oxidation of the zinc mentioned. By the use of electrodes of large cross section, it is probable that this concentrated heating could be reduced, so that rather than a high temperature at one place there would be a uniform comparatively low temperature all over the charge. This was found to be the case in the smelting of iron ores both at Heroult, Cal., and Trollhättan. Or the current could possibly be distributed more evenly by flat electrodes of rectangular cross section. The formation of blue powder is also caused by volatilization of silicon which condenses on the zinc either as silicon or silica. For example, in some experiments at the Bureau of Mines laboratory, the condensed fumes from a ferrosilicon furnace contained 67.69 per cent SiO_2 , 9.72 per cent Al_2O_3 , 4.11 per cent FeO , 0.20 per cent MgO , 1.90 per cent CaO , with the rest carbon. This volatilization of silicon occurs in any electric furnace operated with an arc, as in ferrosilicon manufacture, when there is silica in the charge. At the lower temperature of a resistance furnace it is not so likely to occur.

This problem, although difficult, should be solved in time. When it has been solved and there is no need of resmelting a large proportion of blue powder as at Trollhättan, where 2 tons of blue powder is smelted for each ton of ore treated, it is probable that electric zinc smelting will proceed rapidly. The use of iron as a desulphurizing agent does not seem to have advanced as far as reduction of the oxide with carbon, and it is probable that the latter method will keep its present supremacy. Some work has been done in an electric furnace operated under pressure or vacuum, but only a beginning has been made in this direction.

TIN.

Although rather extensive experiments have been conducted in England on the smelting of tin ores in the electric furnace,^a the authors know of no place where the electric furnace is being used for that purpose at the present time. Moreover, as only small quantities of tin ore have ever been found in the United States or its possessions, the use of the electric furnace for the smelting of tin ores will not be discussed.

^a Harden, J., Electric tin smelting: *Met. and Chem. Eng.*, vol. 9, 1911, pp. 453-457.

PART III. THE MANUFACTURE OF FERRO-ALLOYS IN THE ELECTRIC FURNACE.

By ROBERT M. KEENEY.

INTRODUCTION.

The subjects presented in this part of the report include an account of the development of the manufacture of ferro-alloys in the electric furnace, descriptions of some types of electric furnaces used in ferro-alloy production, descriptions of several European plants that have been visited by the writer, and information regarding the manufacture and uses of each ferro-alloy.

A ferro-alloy is an alloy of iron so rich in some element other than carbon that the alloy is used primarily as a vehicle for introducing that element into the manufacture of iron and steel. Ferro-alloys are not usefully malleable, and they usually contain more of the alloying element than is desirable in an alloy steel.

Ferro-alloys may be used in the manufacture of iron and steel in two ways. First, the ferro-alloy may be added to the steel for its cleansing and deoxidizing effect with the intention that the alloying element will combine with oxygen or some other impurity in the steel bath and pass either wholly or for the greater part into the slag. Second, the ferro-alloy may be used as a fixed addition, the alloying element staying in whole or for the greater part in the steel, to which it imparts some desirable quality.

The problem encountered in the manufacture of ferro-alloys has been the production of an alloy with a low percentage of carbon, a high percentage of the alloying element, and a low percentage of impurities that might be injurious to steel. The alloy should have a low enough melting point so that it can be used in a steel bath of ordinary temperature. To obtain this feature it has been necessary in some cases to produce an alloy with a high percentage of carbon and a comparatively low percentage of the alloying element. The ferro-alloys with a low percentage of carbon are considerably more expensive than the high-carbon alloys, because of the extra refining necessary, which requires a higher power consumption and increased quantities of expensive ores for deoxidization. It is also essential to produce a ferro-alloy of uniform composition, a difficult feature with some alloys. A ferro-alloy that is not uniform is a source of

trouble and expense in the manufacture of alloy steel, because it is not possible to regulate closely the composition of the steel produced. How these points have influenced the development of the manufacture of ferro-alloys is shown in the following discussion.

The most important ferro-alloys in order of their use are ferromanganese, spiegeleisen, ferrosilicon, ferrochrome, silicomanganese, ferrotitanium, silicomanganese-aluminum, ferrotungsten, ferrovanadium, ferronickel, and ferrophosphorus. Other alloys not so widely used but employed by some steel makers are ferrocobalt, ferrosilicon-aluminum, silicocalcium-aluminum, and ferromolybdenum. Other ferro-alloys that have been investigated by the manufacturers but for which no wide use has as yet been found are ferroboron, ferro-uranium, and ferrotantalum.

EARLY DEVELOPMENT OF THE MANUFACTURE OF FERRO-ALLOYS.

DEVELOPMENT PREVIOUS TO THE USE OF THE ELECTRIC FURNACE.

Up to the year 1899, when the electric furnace was first used for the manufacture of ferro-alloys, all such alloys were made in the blast furnace or the crucible furnace. Ferromanganese with a high percentage of manganese was first produced at Bonn, Germany, in 1886.^a This alloy was made in a crucible and contained 70 to 80 per cent manganese. In 1873 the manufacture of ferromanganese was first conducted in the blast furnace, a method by which the greater part is manufactured to-day. In 1875 ferrosilicon containing 10 to 12 per cent silicon was made in the blast furnace. The manufacture of ferrochrome in a commercial way in the blast furnace was started in 1886. Thus up to the year 1890 the only method of manufacture of ferro-alloys was the use of the crucible, the blast furnace, or occasionally the open hearth. In every case carbon in the form of coke or anthracite coal was used as a reducing material.

The production of a ferro-alloy with a low carbon content or a high percentage of the alloying element is limited in the blast furnace by three difficulties—first, the temperature is too low for the reduction of some of the oxides of the alloying metals, so that only alloys of metals reduced at a comparatively low temperature or of low melting point can be made; second, it is difficult to obtain an alloy containing a high percentage of the special metal; and third, it is impossible to produce a ferro-alloy low in carbon because of the great excess of carbon in the charge.

With the crucible, owing to the small scale of operation necessary, the process is necessarily expensive. Owing to the temperature

^a Venator, W., *Über Eisen Liegierungen und Metall für die Stahl Industrie: Stahl und Eisen*, vol. 28, 1908, p. 41; *Iron and Coal Trade Rev.*, vol. 76, 1908, p. 520.

limitation, certain oxides can not be reduced and metals of high melting point can not be melted; it is difficult to obtain an alloy with a high percentage of the special metal; and if a graphite crucible is used, the percentage of carbon tends to be high in the ferro-alloy.

For these reasons, up to 1890 the manufacture of ferro-alloys was mostly confined to the production of ferromanganese of 70 to 80 per cent manganese, ferrosilicon of 10 to 12 per cent silicon, and ferrochrome of 30 to 40 per cent chromium and of 6 to 12 per cent carbon.

INTRODUCTION OF THE ELECTRIC FURNACE.

The opportunity for improvement in the production of high-grade ferro-alloys by the use of the electric furnace was made evident by the researches of Moissan,^a beginning in 1890, upon the reduction of metallic oxides in the electric furnace. Moissan used an arc furnace, consisting of a chamber of refractory material, into which two horizontal carbon electrodes were introduced.

The commercial production of ferro-alloys in the electric furnace began in 1899, when the Bullier patents for the manufacture of calcium carbide were declared valid in France. There was also an overproduction of calcium carbide, which culminated at the same time. The overproduction rendered it necessary for many established carbide works to seek new uses for their electric power and electric-furnace installations. Those works began the manufacture of ferrosilicon, ferrochrome, and ferromanganese in their old carbide furnaces. At that time the manufacture of ferromanganese in the electric furnace did not prove successful because of the high loss by volatilization, and it was abandoned. The production of high-percentage ferrosilicon, ferrochrome, and other ferro-alloys not easily volatilized was very successful.

By 1902 ferrosilicon containing 25 per cent silicon was being used to a considerable extent instead of the blast-furnace alloy of 10 to 12 per cent silicon. This development was quickly followed by the introduction of ferrosilicon containing 50 per cent, 75 per cent, and 90 per cent silicon. The high-grade ferrochrome of 60 to 70 per cent chromium content and 5 to 8 per cent carbon rapidly replaced the blast-furnace product of 30 to 40 per cent chromium and 6 to 12 per cent carbon. Later the production of ferrochrome containing less than 1 per cent carbon was begun. As the refining possibilities and high temperature available in the electric furnace were more fully appreciated, the production of many ferro-alloys not previously made began. Also in recent years, by careful control of the temperature of the electric furnace, which is possible in the ordinary arc furnace when run as a resistance furnace, it has been found possible to make ferromanganese without an excessive loss of manganese by volatilization.

^a Moissan, H. The electric furnace. Translated by V. Lenher, 1904.

This ferromanganese is usually combined while molten with molten ferrosilicon to produce silicomanganese, or a similar alloy.

There has been comparatively little change in the type of electric furnace used for making ferro-alloys since 1899. In many plants the old carbide furnaces are still in operation, and where the ferro-alloy is simply being produced directly from the ore with carbon as a reducing agent, the furnace is essentially of the old Siemens crucible type, having a conducting hearth of carbon and an upper vertical electrode (fig. 38). There are several modifications, which are described later, but their difference in design is mainly in shape and in the manner of introducing the current. With one or two exceptions, introduced lately, ferro-alloy furnaces are run with open tops, as in the first years of manufacture. There is no shaft, so that most of the reduction is by solid carbon.

PRESENT STATUS OF THE MANUFACTURE OF FERRO-ALLOYS IN THE ELECTRIC FURNACE.

The growth of the ferro-alloy industry in Europe has been rapid since 1899, but comparatively slow in the United States. In Europe there are about 25 plants engaged in the manufacture of ferro-alloys by the electric furnace, as compared with two plants in the United States. In addition to these there are three companies using means other than the electric furnace in the United States, and one electric-furnace ferrosilicon plant in Canada.

There are several reasons why the growth has been slower in America than in Europe. Hydroelectric power is not as cheap here and not so favorably located for the receipt of raw material and the sale of the product. The water-power sites can not be developed so cheaply as many of the foreign sites, where the cost per kilowatt-year varies from \$10 to \$20, as compared with \$20 to \$40 in the United States. In Canada power is somewhat cheaper, but much of it is in inaccessible places. Most of the Norwegian and Swedish plants are on tidewater or navigable rivers. French works are within about 200 miles of Marseilles. The use of ferro-alloys in the manufacture of high-class steels did not advance as rapidly in the United States as in Europe. And owing to less favorable natural advantages, electrochemical and electrometallurgical industries in general have not made so rapid progress and growth.

A large proportion of the ferro-alloys used in the United States is imported, as, although there is a duty, local manufacturers are not able to supply the whole demand. About one-half of the ferromanganese and one-half of the ferrosilicon, as well as a large part of the ferrotungsten, used in the United States is imported. More ferrotitanium and ferrovanadium is manufactured here than abroad. The ferrochrome production just about supplies the domestic demand.

In Europe the industry of manufacturing ferro-alloys in the electric furnace is in excellent condition commercially, with the demand for alloys steadily increasing. Because of the race between European countries in the building of large navies, there is a great demand for ferrochrome. In Europe, the sale of ferro-alloys, especially ferrosilicon and ferrochrome, is controlled by a syndicate of the various plants, each plant, according to its capacity, supplying a certain part of the total market demand.

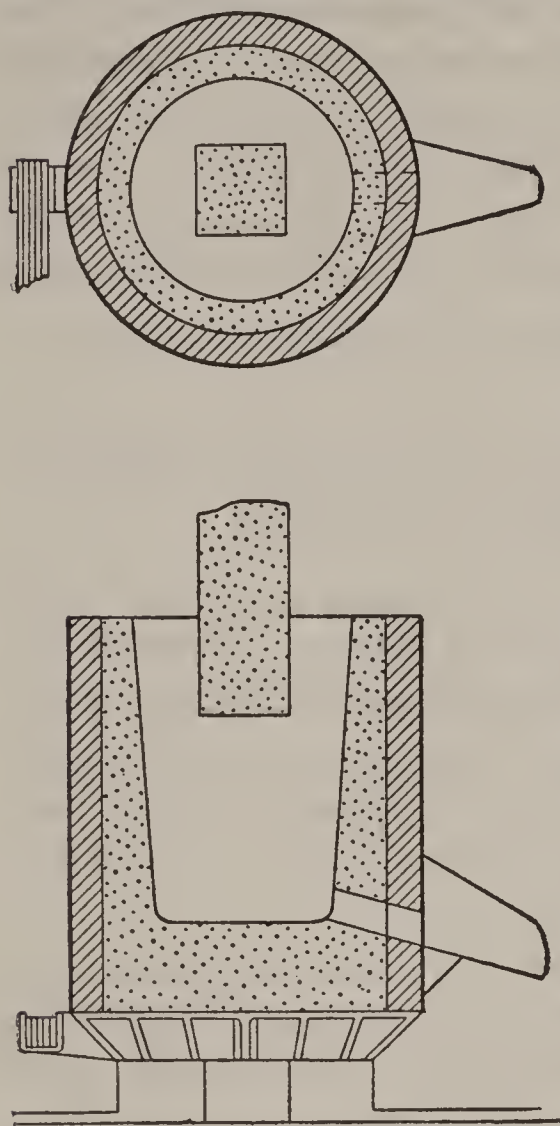


FIGURE 38.—Plan and elevation of Siemens type of ferrosilicon furnace.

In the United States there is a ferro-titanium manufacturing company at Niagara Falls, N. Y. An electrometallurgical company has one plant at Kanawha Falls, W. Va., where the principal product is ferrochrome, and another at Niagara Falls, N. Y. Other alloys made are ferrosilicon, ferrotungsten, ferrovanadium, ferromolybdenum, and ferrophosphorus. A chemical company at Primos, Pa., manufactures metals and ferro-alloys by chemical methods or in combustion furnaces. Among its products are ferrotungsten, ferrovanadium, tungsten metal, ferromolybdenum, ferrochrome, ferronickel, and ferroboron. Another company manufactures ferrovanadium by the thermit process at Bridgewater, Pa. Another company has a plant for the manufacture of metals and ferro-alloys at Newark, N. J., but

imports most of its products from its foreign works. Among its products are ferrotitanium, ferrovanadium, ferromolybdenum, ferrochromium, and chromium.

TYPES OF ELECTRIC FURNACES USED IN FERRO-ALLOY MANUFACTURE.

FERROSILICON FURNACE OF THE SIEMENS TYPE.

A furnace commonly used in the manufacture of ferrosilicon and other ferro-alloys is of the Siemens type. (Fig. 38.) The furnace is circular in cross section with one upper carbon electrode and a conducting hearth of carbon. The electrodes are held by holders of various types, some of which are shown in figure 39 and in figures 17 to 35. The holders are attached to cables running to hand adjustment wheels. The electrode holders are not usually water cooled.

As yet the use of round electrodes threaded for continuous feeding has not been widespread in ferro-alloy works. In the ferrosilicon furnace it is necessary to line the bottom with carbon paste resting on an iron plate to which the bottom electrical connection is made. The paste of carbon may be surrounded by fire bricks, which are set in a steel shell or are held together by sheet-iron bands. The fire bricks are often not used at all. There is one tap hole. Fire brick can not be used as the interior lining in a ferrosilicon furnace, because it is rapidly attacked by the ferrosilicon. The older furnaces are operated with single-phase alternating current, but the more recent installations are connected to a phase of a three-phase circuit as shown in the lower view of figure 50. These furnaces are built in sizes vary-

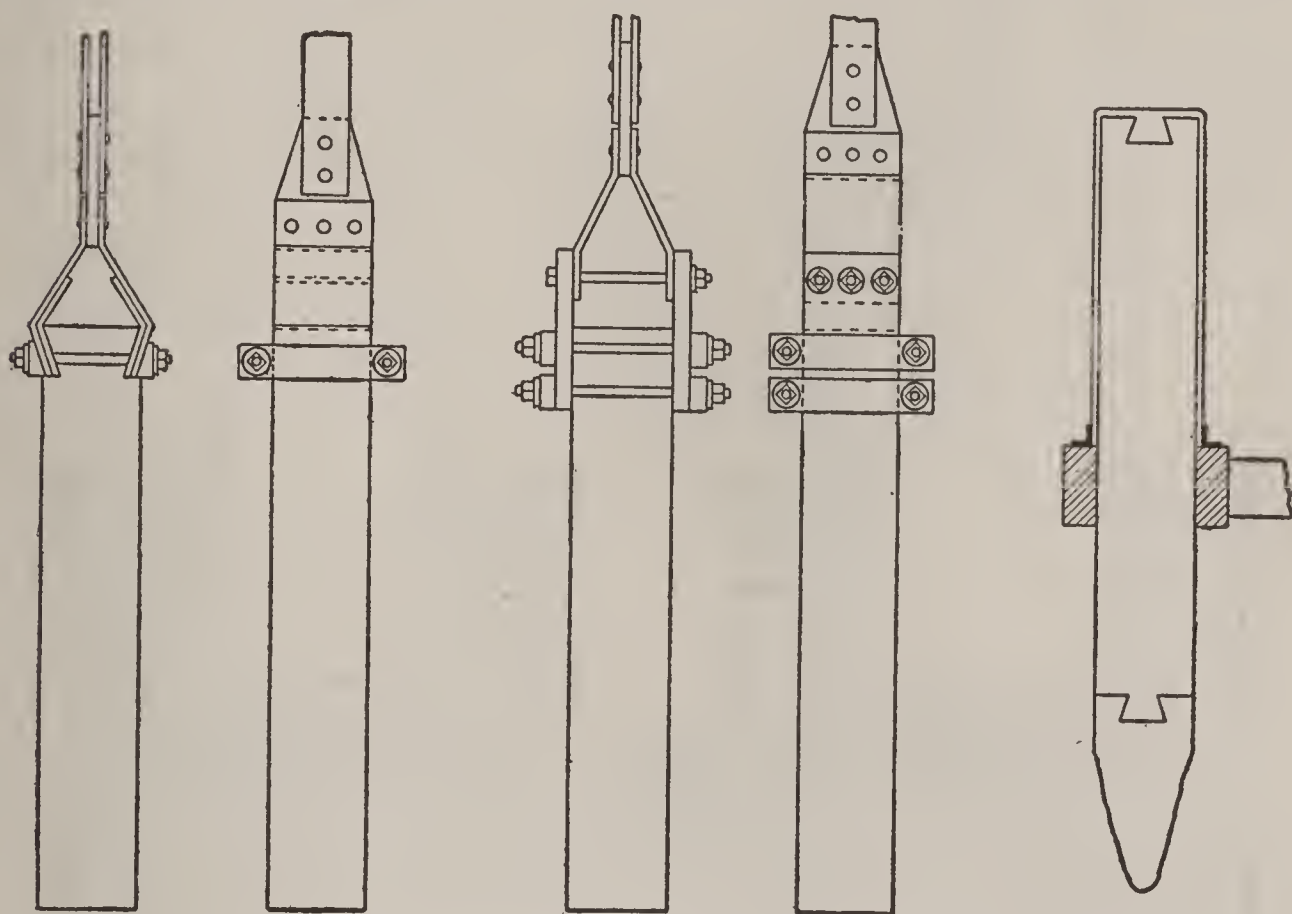


FIGURE 39.—Electrodes and holders used in Siemens type of ferrosilicon furnace.

ing from 200 to 1,000 kilowatts. A 225-kilowatt furnace may have a crucible 2 feet 6 inches in diameter at the top and about 2 feet in diameter at the bottom. The external dimensions are: Diameter, 4 feet; height from base to top, 5 feet; masonry base, about 1 foot 3 inches high. The lining is about 9 inches thick. The furnaces are operated at 50 to 80 volts. A 12-inch square or circular electrode is used. The power factor varies from 0.8 to 0.9.

If a ferrochrome or alloy of medium carbon percentage other than ferrosilicon is to be made in a furnace of this type, the carbon bottom is used, but in the walls the carbon is replaced by a basic lining of magnesite and tar. After a furnace has been operating some time, the material of the charge chills on the walls forming its own lining, and some of the ferro-alloy reduced tends to freeze on the hearth, thus protecting the main part of the molten metal from the

carbon. If a ferrochrome containing less than 2 per cent carbon is desired, the lining is of chromite, but this is rather expensive. Ferro-silicon and ferrochrome furnaces of this type have been in continuous operation for periods of two to four years.

Usually the smaller furnaces are charged by a laborer, who shovels the new materials from the floor at the tapping level into the top of the furnace. Larger furnaces have a steel platform around the top for charging from a bin system. In some works the furnaces are set on trucks, so that an old one may be quickly replaced by a new one.

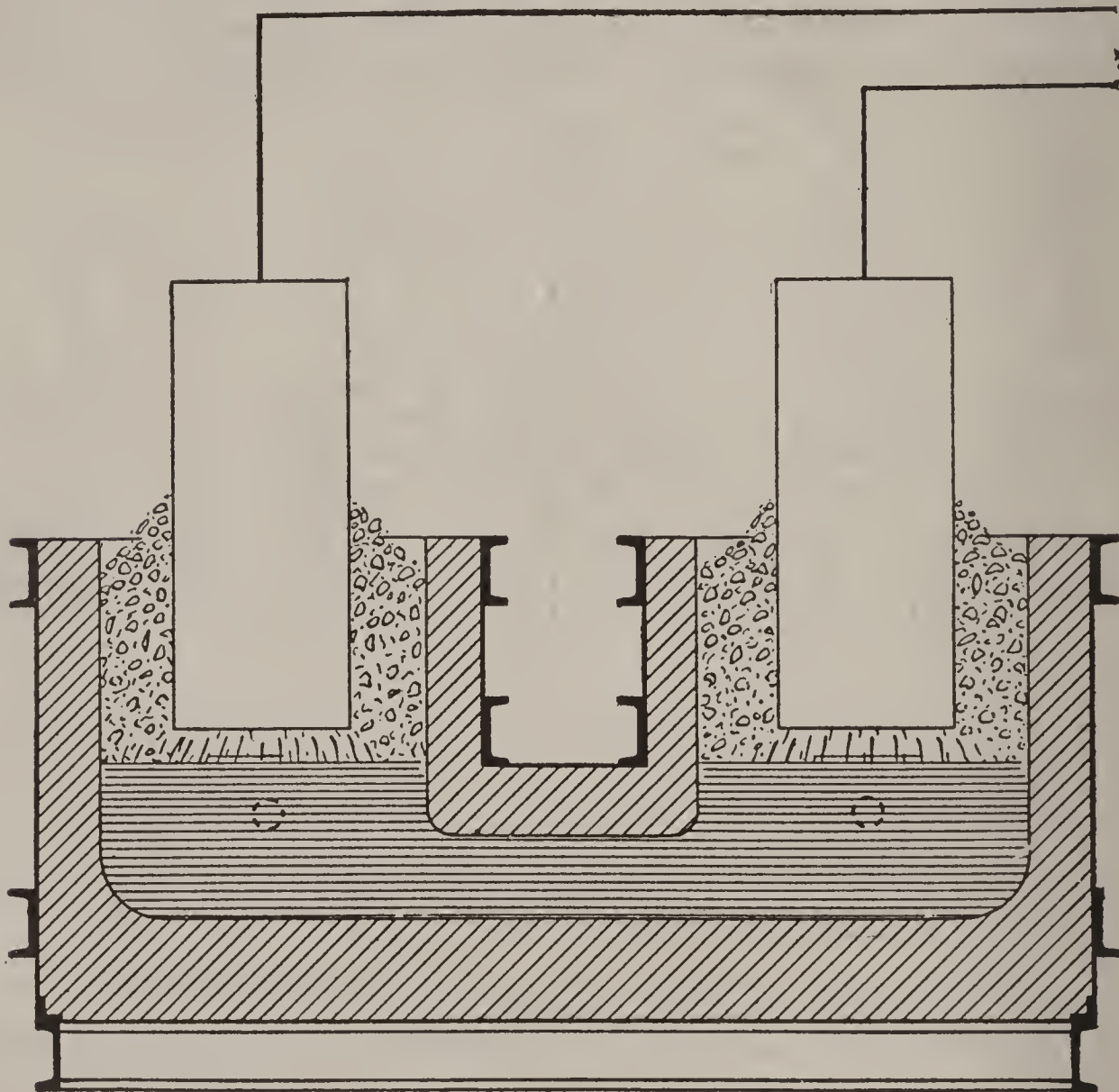


FIGURE 40.—Keller ferro-alloy furnace.

KELLER FURNACE.

The Keller furnace (fig. 40), used for the manufacture of calcium carbide and ferro-alloys, consists of two iron casings of square cross section, forming two shafts that connect with each other at the lower end by means of a lateral canal. There are two vertical carbon electrodes connected in series with the slag and the metal. Ordinarily the current flows through the electrode in one shaft through the charge and through the other electrode, but when the furnace is being started after repairs the connection between the electrodes is

made by carbon blocks at the base of each shaft, which are connected by copper bars. There are two tap holes, one in each shaft. The lining used depends on the alloy being made. The electrodes are built of carbon blocks to a total cross section 30 inches square, and are not designed for continuous feeding. They are 4 to 5 feet long. The furnaces are about 14 feet long, 7 feet high, and 6 feet wide. Extra electrodes are kept ready to replace worn ones. Adjustment of electrodes is by hand.

The Keller furnace is built in sizes using 900 to 2,200 kilowatts. Some furnaces are connected as shown in the lower view of figure 50 and others are directly connected to a single-phase system. The furnaces are operated at 50 to 70 volts. The power factor is about 0.7 to 0.8.

CHAPLET FURNACE.

The Chaplet furnace (fig. 41) for ferro-alloy manufacture has two electrodes in series, one of which is in one chamber, the other being on a channel extending from the main smelting chamber. The current passes through a vertical carbon electrode, through the bath and channel to the other vertical electrode.

The Chaplet furnace has been used by one company especially in the manufacture of low-

carbon ferro-alloys. In making low-carbon alloys it is usually operated with a roof when refining, but when reducing directly to a high-carbon alloy the roof is removed. The lining is of dolomite, magnesite or chromite, with a silica-brick roof. The furnace has been built in sizes of 120 to 150 kilowatts.

GIROD RESISTANCE CRUCIBLE FURNACE.

In its early experiment work the Girod Co. devised a crucible furnace in which heating is by resistance. This furnace (fig. 42) consists of a crucible of some refractory material, which is surrounded

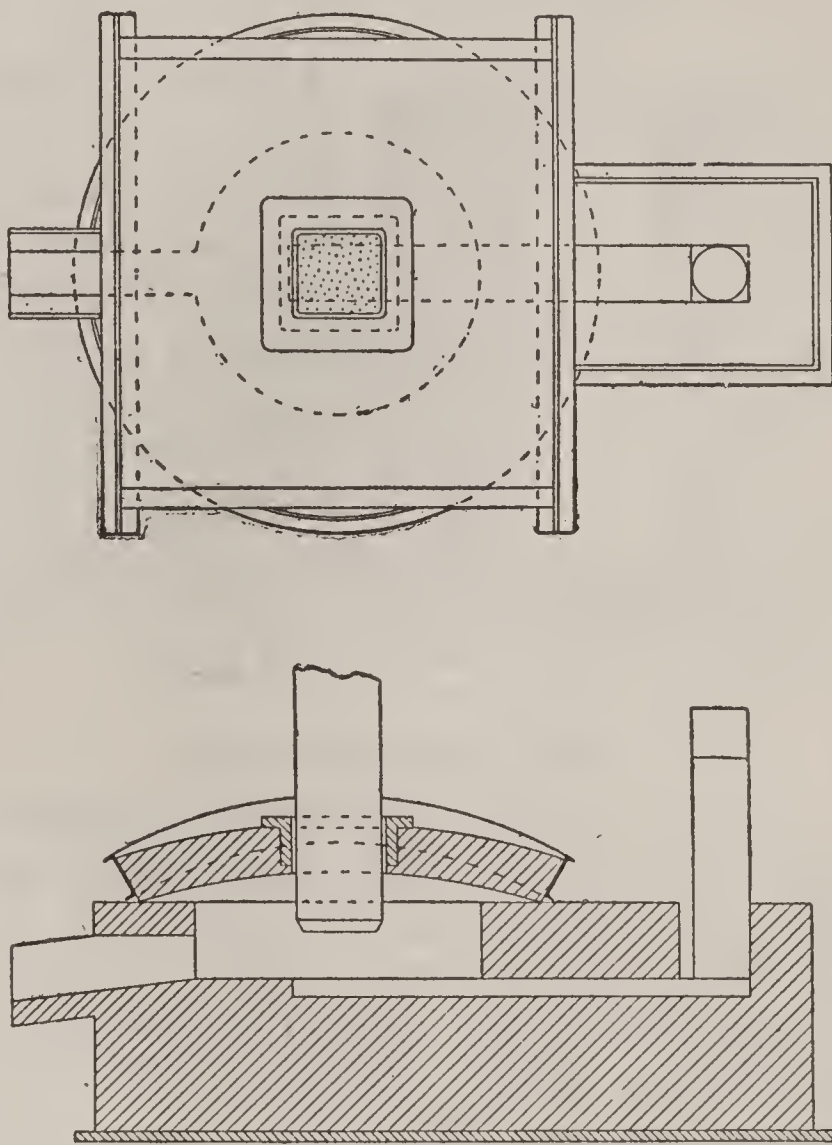


FIGURE 41.—Plan and elevation of Chaplet ferro-alloy furnace.

by a material of high electrical resistance. The use of ferrosilicon for this purpose was patented by Girod. Sometimes coke is mixed

with the ferrosilicon to form the resistor. Outside of the resistor are set four carbon blocks, which serve as electrodes and are separated by brick of high insulating qualities to prevent short circuits. The whole is incased in a steel shell and mounted on a framework for tilting by hand. The materials are charged in by hand, and the crucible is covered with a brick cover. Either direct or single-phase current may be used. A general difficulty with a furnace of this type is the prevention of short circuits caused by uneven heating of the crucible, and care must be taken to connect the furnace so that the current takes the longest possible course. The furnace is used for manufacturing ferro-alloys that do not require a temperature above $1,600^{\circ}$ C. It is built in sizes up to 150 kilowatts.

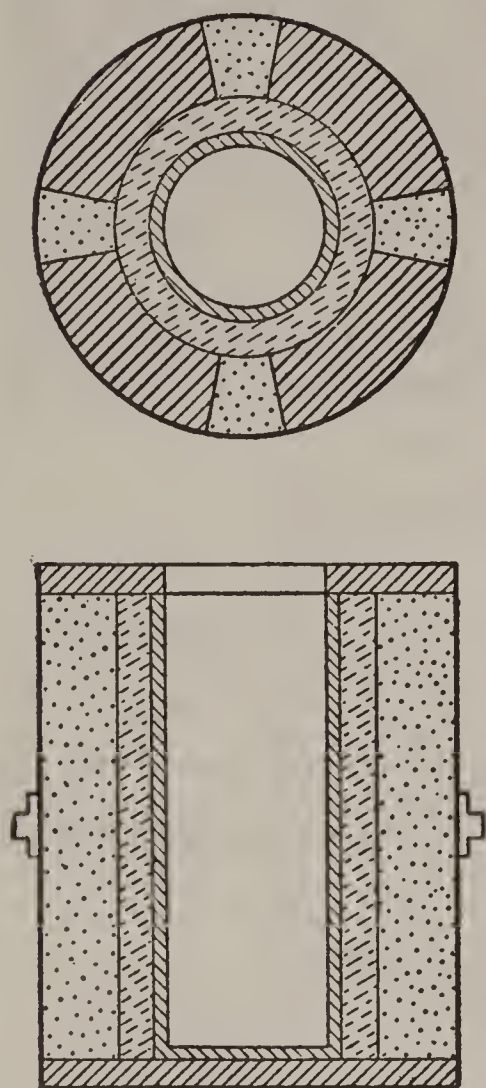


FIGURE 42.—Plan and elevation of Girod resistance furnace.

GIROD ARC FURNACE.

For the manufacture of ferro-alloys of high reducing or melting point, other than ferrosilicon, Girod uses a furnace (fig. 43) of circular cross section, with one or more upper carbon electrodes in parallel and a conducting hearth of dolomite and tar having mild-steel rods embedded or openings into which molten metal is poured and allowed to freeze. These rods are placed radially around the bottom. The ends project outside and are water-cooled where the cables are attached. The lining is usually of dolomite or magnesite throughout, but with the production of low-carbon ferro-chrome a chromite lining may be used. Where there are several upper electrodes the diameter of the furnace is greater and the depth about the same as in the single electrode furnace. The general dimensions are about the same as given for the Siemen's furnace, with a conducting hearth of carbon. Either direct or alternating current may be used.

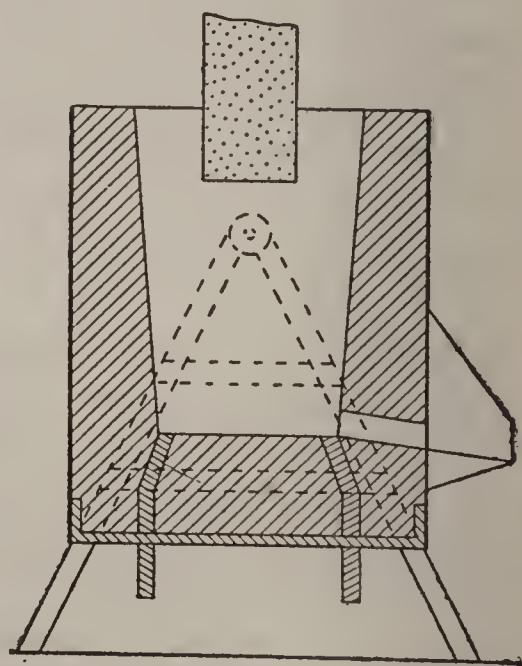
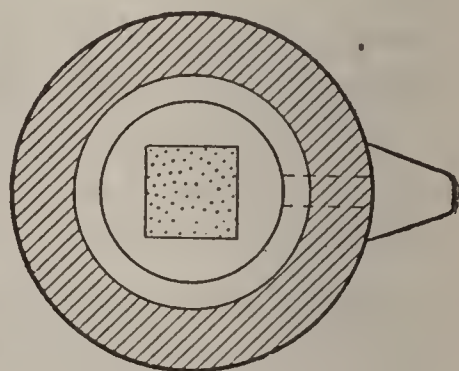


FIGURE 43.—Plan and elevation of Girod electrode ferro-alloy furnace.

MERAKER FURNACE.

At Kopperaaen, near Meraker, Norway, a furnace (fig. 44) similar to the Alby carbide furnace is being used in the production of ferro-alloys, especially ferro-chrome. This furnace is rectangular in cross section, with an upper rectangular carbon electrode, and a conducting bottom of carbon, forming the other electrode. The carbon bottom rests on a concrete foundation or the whole furnace may be set on wheels so as to be movable. The lining consists of magnesite and tar, 9 inches thick. The whole is incased in a $\frac{5}{8}$ -inch steel shell. In one end of the furnace there is a tap hole. At the opposite end connection is made to the bottom electrode. The upper electrode consists of four carbon blocks 10 by 12 inches by 6 feet long, set side by side so as to make an electrode 10 by 48 inches in section.

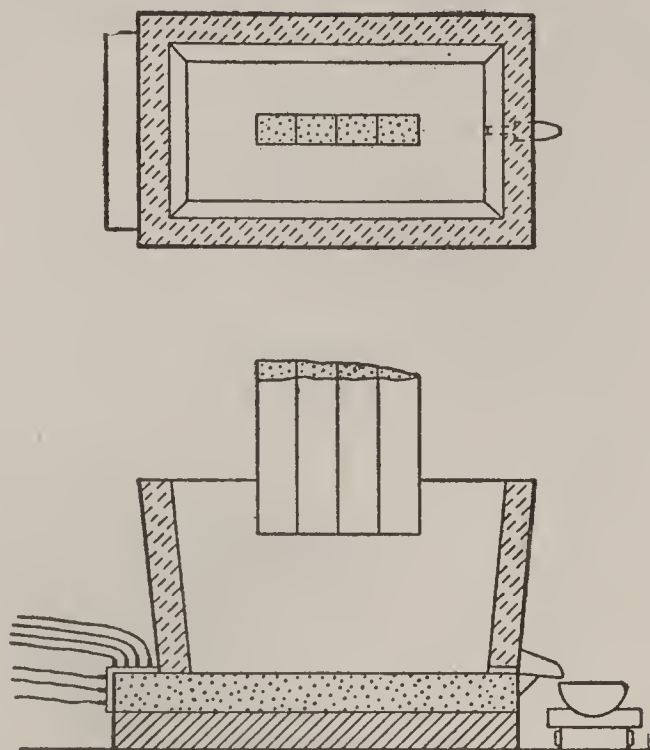


FIGURE 44.—Plan and elevation of ferrochrome furnace at Kopperaaen, Norway.

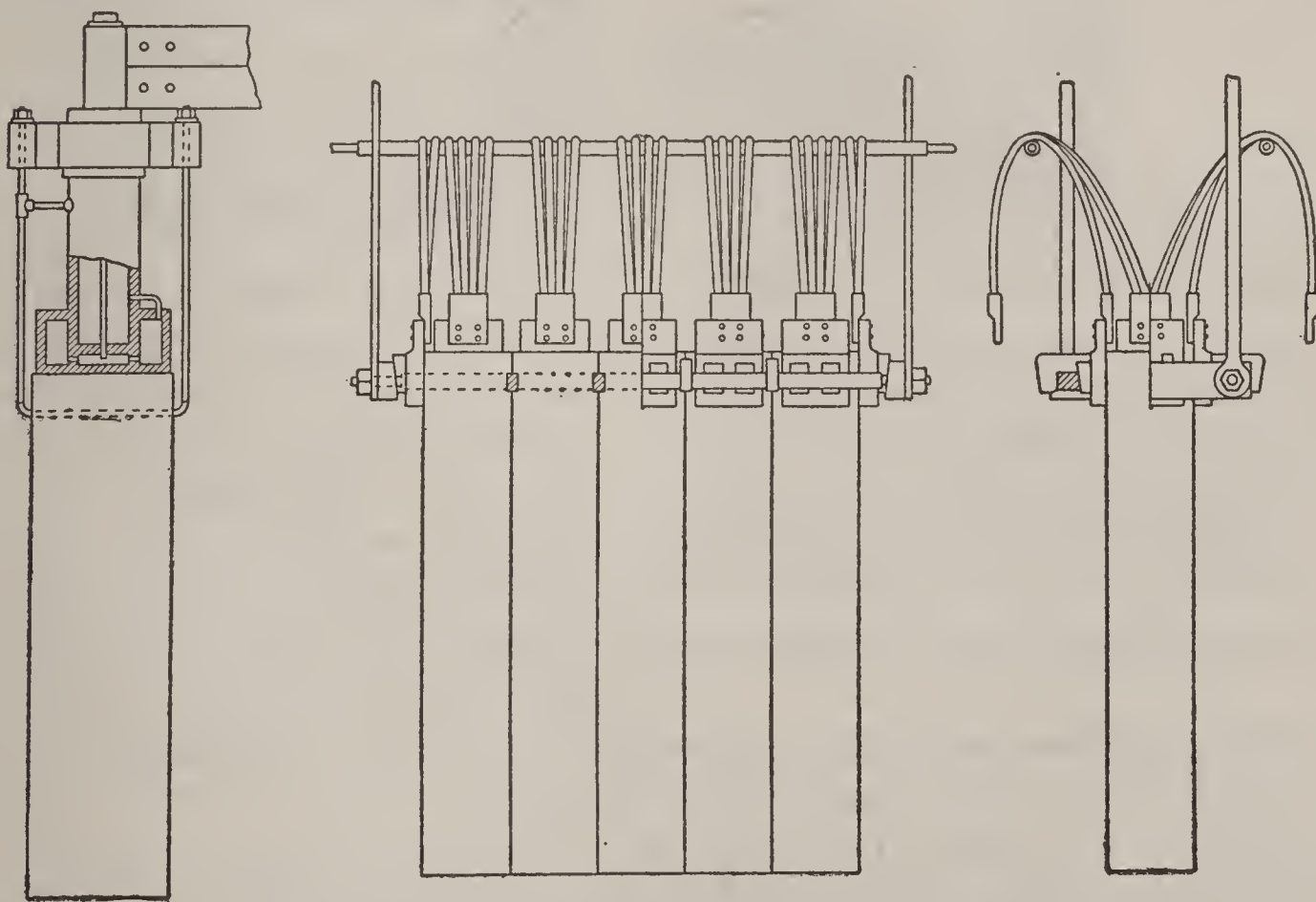


FIGURE 45.—Norwegian and Italian electrodes and holders.

The whole electrode is covered with sheet iron in an endeavor to reduce oxidation and is held by the holder shown in figure 45 (right-

hand view), which is supported by cables from above. A spare electrode is left hanging from the framework above the furnace ready to replace one worn out.

A 750-kilowatt furnace is about 10 by 6 feet in section outside at the top and about 9 feet by 5 feet 6 inches at the bottom. The internal

depth is about 5 feet. The carbon block at the bottom is 1 foot thick and rests on a concrete foundation 1 foot above the floor. The furnace top is about 7 feet above the floor. These furnaces are operated at 65 volts with either direct or alternating current.

A furnace of this design with a closed top has been designed at Kopperaaen. This consists chiefly of a steel hood $1\frac{1}{2}$ feet high, lined with fire brick. There is an opening for the electrode, flues for drawing off the gases, and charging doors. Little gas escapes from this furnace except through the flues. The top reduces the electrode consumption considerably.

SERIES FURNACE.

A furnace designed to avoid the use of a conducting hearth and commonly called the "series" furnace, is shown in figure 46. The furnace shown has two vertical carbon electrodes connected in series for operation with single-phase current. If three-phase current is

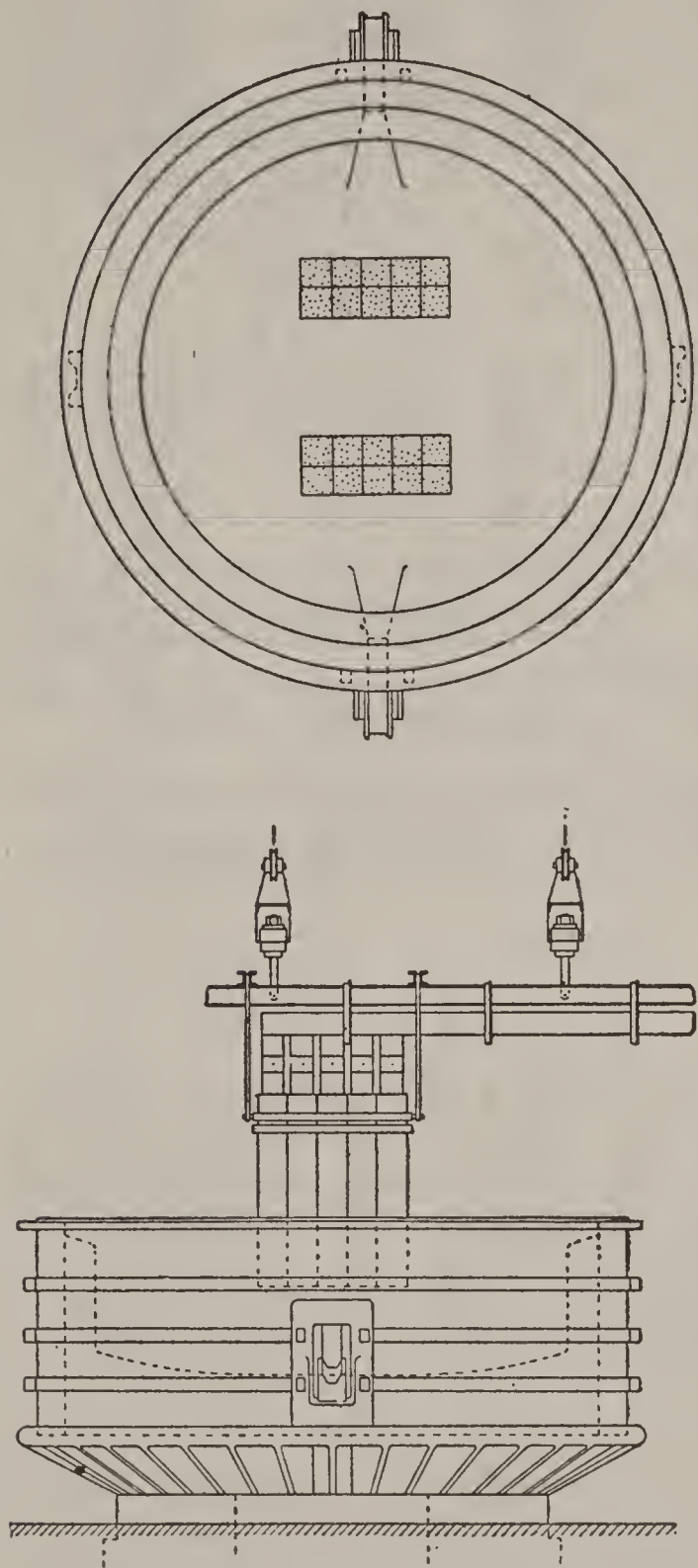


FIGURE 46.—Plan and elevation of single-phase series ferro-alloy furnace.

used there are three electrodes connected each to a phase of the system in the delta connection. The construction of the furnace proper is in general the same as for furnaces already described, except that there is no electrical connection with the hearth. Furnaces of this type have been built in sizes up to 3,000 kilowatts.

Three-phase furnaces may be of the shape shown in figure 46, with three electrodes arranged in a triangle and each connected to a phase

of a three-phase system. Or the furnace may be rectangular, as shown in figure 47. In neither form is there electrical connection to the bottom. A furnace of this type has been built to take 5,000 kilowatts.

HELFENSTEIN FURNACE.

A furnace recently designed for the manufacture of calcium carbide and ferrosilicon on a large scale is the Helfenstein furnace (fig. 48), which has several shafts set side by side feeding a crucible beneath. Each shaft has its own vertical top electrode, but all are connected electrically by bottom electrodes in the form of a conducting bottom common to all the shafts.^a There is no other connection between the shafts, so that it is possible to produce different products simultane-

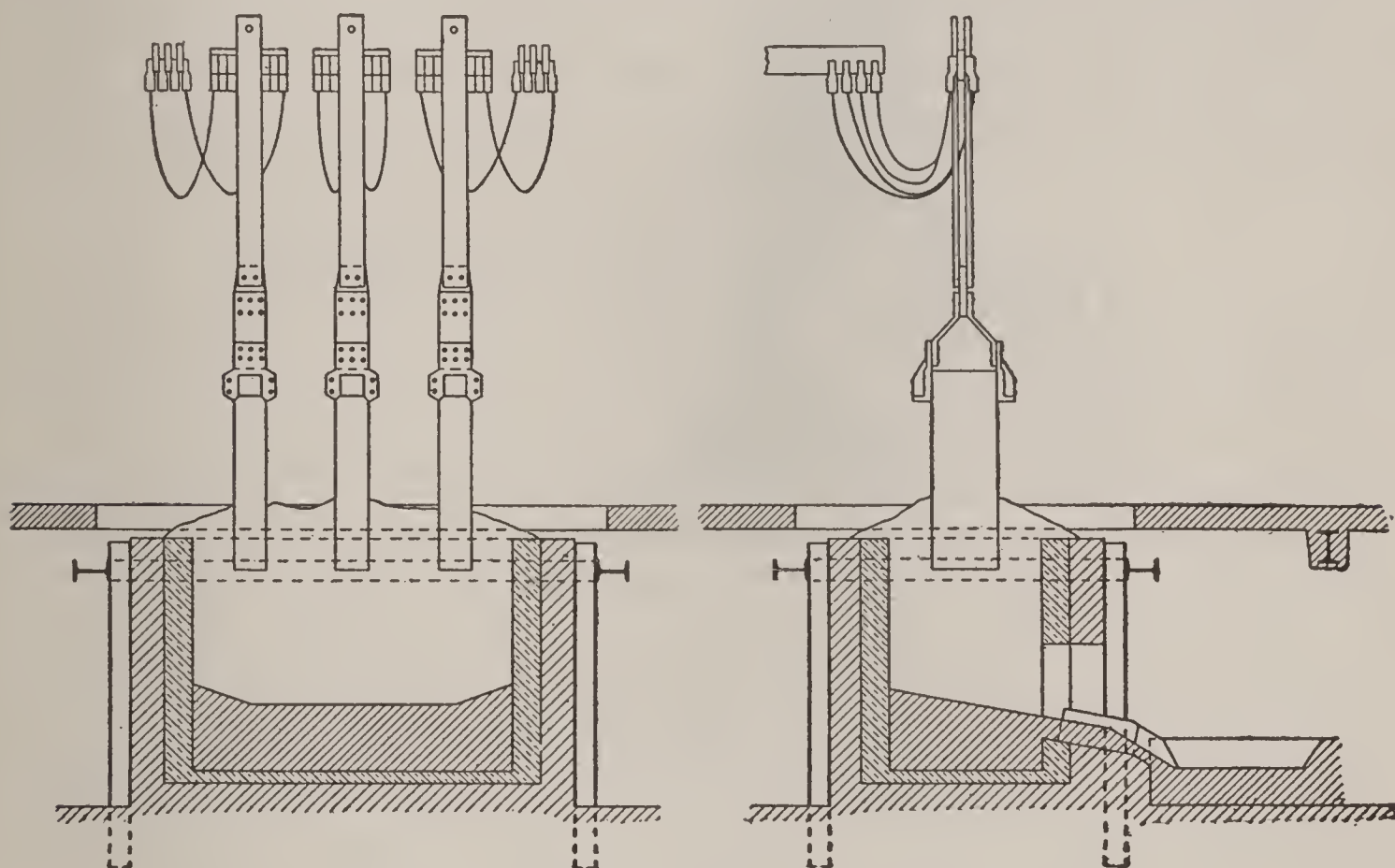


FIGURE 47.—Elevations of three-phase series ferro-alloy furnace.

ously. As shown in figure 48, the shafts enter the crucibles from each side, with the electrodes in the center. The whole furnace is thus rendered air tight. One large 7,500-kilowatt Helfenstein furnace is operating in France and several are being constructed. The use of large units with an air-tight furnace reduces the electrode and the power consumption considerably.

GENERAL CONSTRUCTION OF ELECTRIC FERRO-ALLOY FURNACES.

FUNDAMENTAL PARTS.

From the types of electric furnaces described it may be seen that in general an electric furnace for ferro-alloy manufacture consists of a crucible without a roof, having either a conducting or nonconduct-

^a Taussig, R., Present status of the development of large electric furnace: Met. and Chem. Eng. vol. 10, 1912, p. 686.

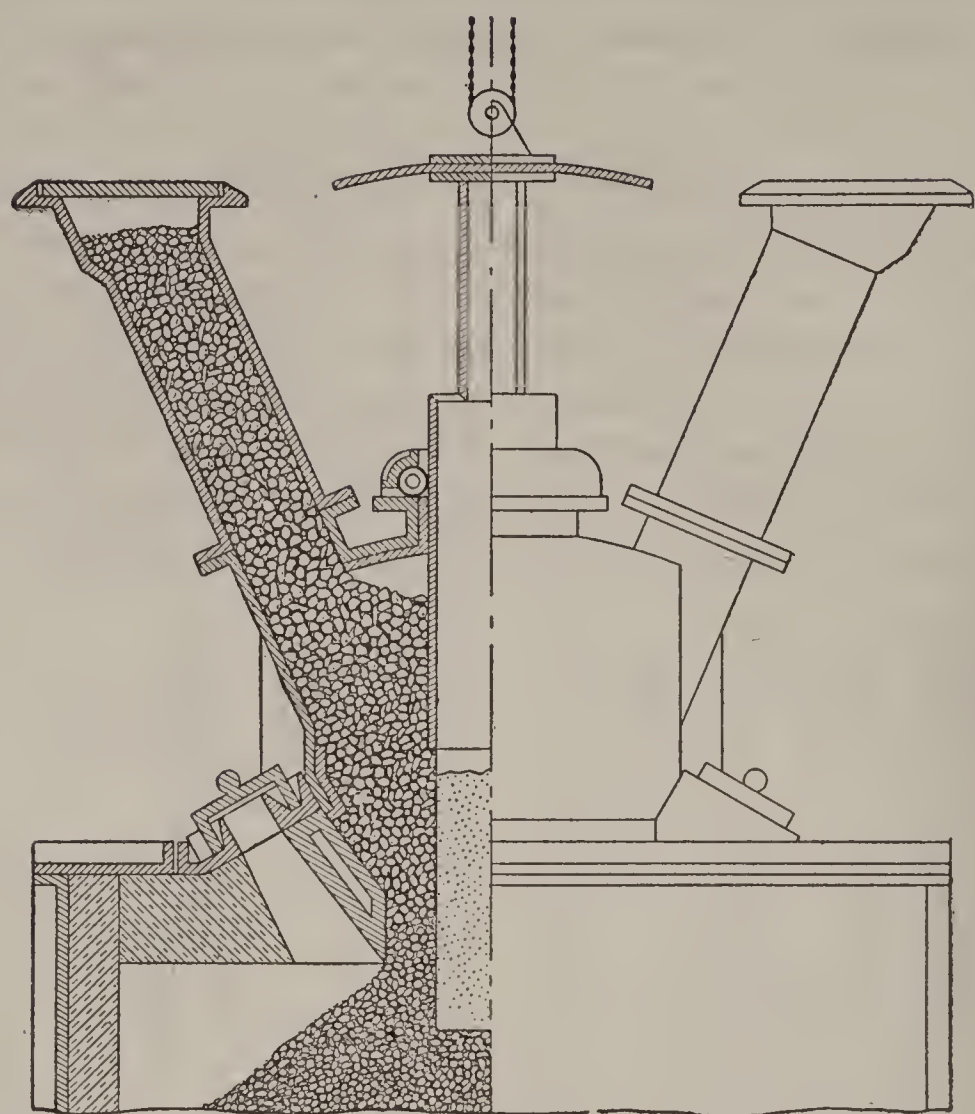


FIGURE 48.—Elevation of Helfenstein furnace.

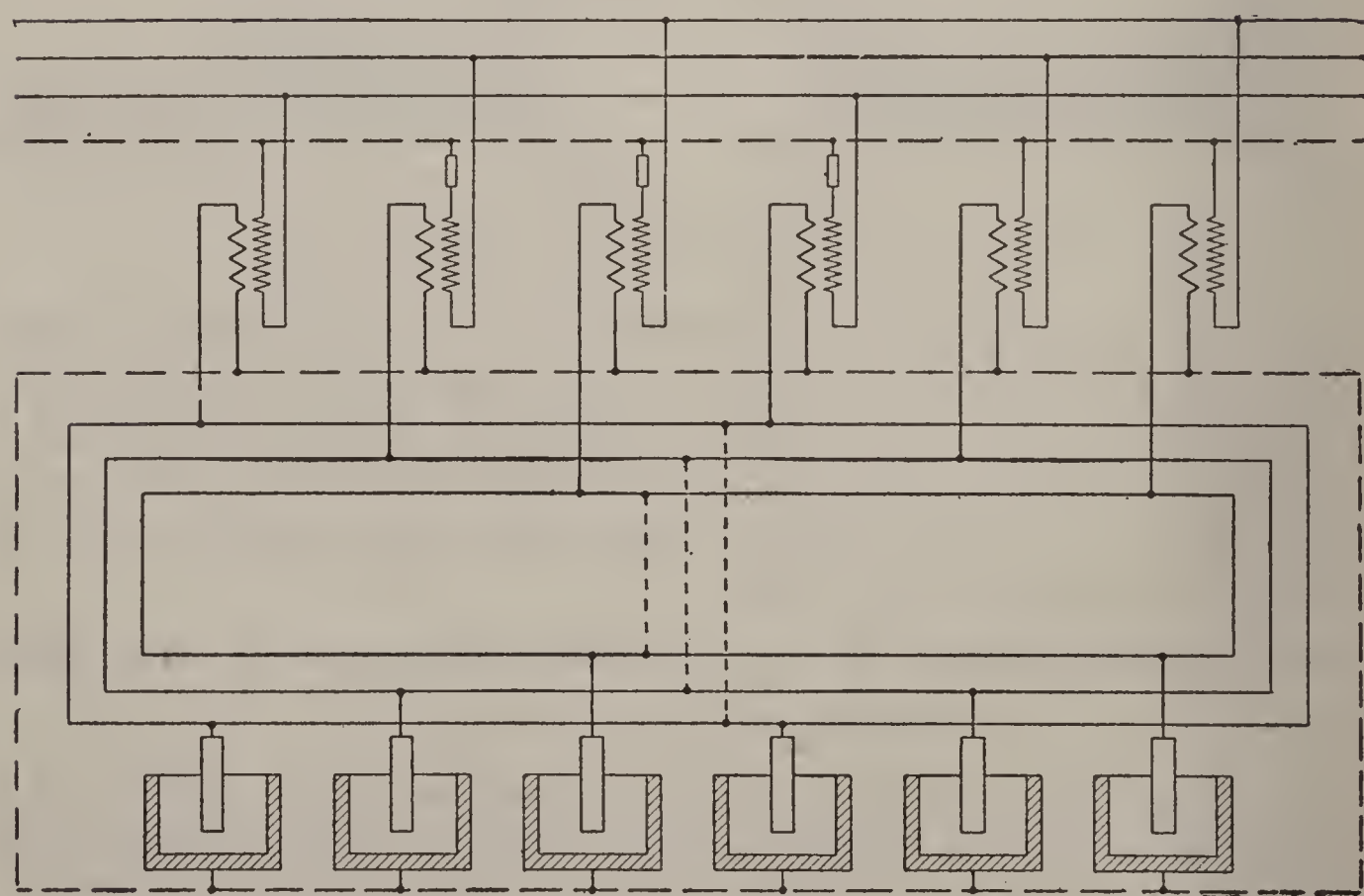


FIGURE 49.—System of electrical connections for several small furnaces.

ing hearth and an upper vertical carbon electrode or electrodes. Most of the furnaces first built had a conducting hearth, and this is still common in furnaces up to about 750 kilowatts, but for larger sizes experience has shown that the nonconducting hearth is preferable. There is a considerable loss of electrical energy by dissipation through the lining, and there is a tendency to have a lower power factor because of the circuit being more completely inclosed in iron. In working a small furnace intermittently—that is, discharging it completely before charging it again—the conducting-hearth furnace is much more easily operated and started when cold. When a large furnace is to be operated continuously for several months at a time on one alloy, this advantage of the conducting hearth is of no importance. A factor little considered in the construction of ferro-alloy furnace is the power factor. Most of the furnaces are completely inclosed in sheet steel, whereas bands would do as well, with much less induction in the circuit.

CONSTRUCTION AND LIFE OF LININGS.

As has been stated, the ferrosilicon furnaces are lined with carbon, and most of the furnaces to be used in producing other alloys with a basic lining of magnesite or dolomite brick or a mixture of tar with either material. When low-carbon ferrochrome is being made, the lining in some furnaces is chromite. Generally a furnace is designed so that the electrode or electrodes are at a sufficient distance from the walls to permit a chilling of the material charged upon the walls. This practice has given the most satisfactory lining. With such an arrangement furnaces have been run two to three years without being relined. The average furnace operating continuously runs about two years before being shut down for repairs. A furnace being operated intermittently must of course be relined oftener.

ELECTRODE HOLDERS.

TYPES.

There are two general types of electrode holders used—first, one with two clamps bolted around the electrode, leaving the top of the electrode exposed; and second, one that grips from above, making it impossible to add a section of electrode from above. The former type has been commonly used in electric steel furnaces, but does not seem to have been so widely used in ferro-alloy manufacture. This is probably because, with the lack until rather recently of an amorphous-carbon electrode threaded for continuous feeding, the most convenient mode of suspension has been by chain with the second type, rather than by a lateral steel frame, as is necessary in the first type. In the future the use of the first type of holder will probably increase because

of the possibility of reducing the electrode consumption by using up all butts. A method formerly sometimes used for joining electrodes is shown in figure 39. This connection was not entirely satisfactory and was not widely adopted.

Various designs of the second type are shown in figure 39. Above the point where it grips the electrode the holder is attached to a chain or cable, by means of which it is supported. With a holder of this type about one-third of the electrode is not used. Electrical connection is made by cables attached to the holder.

In figure 45 is shown a type of electrode used in Norway and Sweden for rectangular furnaces. It consists of carbon blocks held together in a framework of steel by wedges, the electrical connections being made by cables attached to the holder itself. The whole is suspended by cables. In figure 45 (left-hand view) is also shown an Italian method, in which the electrical contact is made against the end of the electrode instead of on the sides for the purpose of getting a more even distribution of current throughout the electrode. It is water cooled and the most perfect design technically, but practically is not very widely used.

In many plants, before any of these holders are connected, each electrode is covered with sheet iron to reduce the electrode consumption by oxidation. At some works it is stated that it is doubtful whether this practice results in enough saving to warrant the additional expense. The built-up electrodes are set in a cement of tar and carbon to fill the cracks.

Great care must be taken in building up the electrodes or there will be excessive local heating.

WASTE OF HEAT IN OPEN-TOP FURNACES.

As most of the reduction of oxides in ferro-alloy manufacture is done by solid carbon at a high temperature, most furnaces are run with an open top into which the charge is placed, around the electrode. There results a considerable waste of heat that might be used in preheating the charge. In most of these furnaces the electrode dips about 12 to 15 inches beneath the top of the furnace, which is kept filled with cold material so that the molten mass beneath will not be exposed. There is burning of carbon monoxide around the electrode and an escape of a large volume of gas through the top of the furnace house. As previously mentioned, there has been a seemingly successful attempt made to overcome this waste at Kopperaaen, Norway, with a modification of an existing furnace. The Helfenstein furnace has this objection removed as it is air-tight, and in addition has feeding shafts, where the charge is preheated by the waste gases. In the manufacture of the higher-priced alloys, such as ferrotungsten and ferrovanadium, it is impossible to use any

furnace but one with an open top because of the intermittent nature of the operation, but in making ferrosilicon and ferrochrome, it seems advisable to preheat the charge as much as possible.

CHARACTER OF ELECTRIC CURRENT.

A large number of the first ferro-alloy furnaces were supplied with direct current when the furnace house was built adjoining the power house so that there was only about 20 feet between the furnace and the generator. It was soon found that the electrolytic action in a direct-current furnace resulted in deposition of potassium, sodium, calcium, aluminum, and other metals with the alloy so that as pure a product could not be obtained with direct as with alternating current. When alternating current was first used, single-phase current was more common, so that the arrangement remained the same as previously, the generator delivering directly to the furnace at the desired voltage without any transformation.

Later when three-phase current became common, furnaces designed to take single-phase current were connected in groups to three-phase systems by several methods. A method commonly used for the furnace with a conducting hearth and one or more vertical electrodes is shown in figure 49 (p. 114). The primary and secondary are both connected in star connection, with the furnace bottom as the neutral point of the secondary circuit. There are two transformers to a phase, supplying two furnaces. In this system there is apt to be an unbalancing of the load on the three phases. It is used only with small furnaces. To get a better balancing of phases, conducting-hearth furnaces are connected as shown in figure 49, where each furnace is connected to a phase or there may be two furnaces to a phase, the whole system being in delta connection.

For single-phase furnaces the method shown in figure 50 is the best, where three nonconducting bottom furnaces are each connected by the delta arrangement to the three transformers of a three-phase line. Thus the primary and secondary are both connected by the delta system. By this arrangement there is a better balancing of the load in the separate phases. Also it is possible to cut out one furnace if desired.

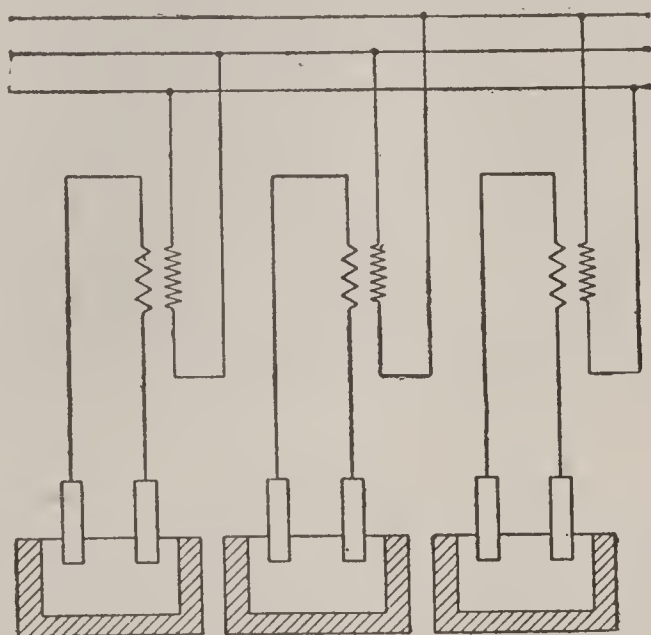


FIGURE 50.—Delta connections of three single-phase furnaces.

A three-phase furnace with a nonconducting bottom is connected as in figure 51, where three electrodes are each connected to a transformer, there being a common neutral line between the three transformers. It is not difficult to keep the phases balanced in this system, and it is very satisfactory. If there is more than one electrode to a phase in a furnace of this type it has been found difficult to keep the phases balanced. That has been the experience with one electric pig-iron furnace, whereas with others no difficulty has been experienced.

SOME EUROPEAN ELECTRIC-FURNACE FERRO-ALLOY PLANTS.

KELLER, LELEUX & CO., LIVET, ISÈRE, FRANCE.

The main works of Keller, Leleux & Co. are at Livet, Isère, France, about 20 miles from Grenoble. There is also a small plant using

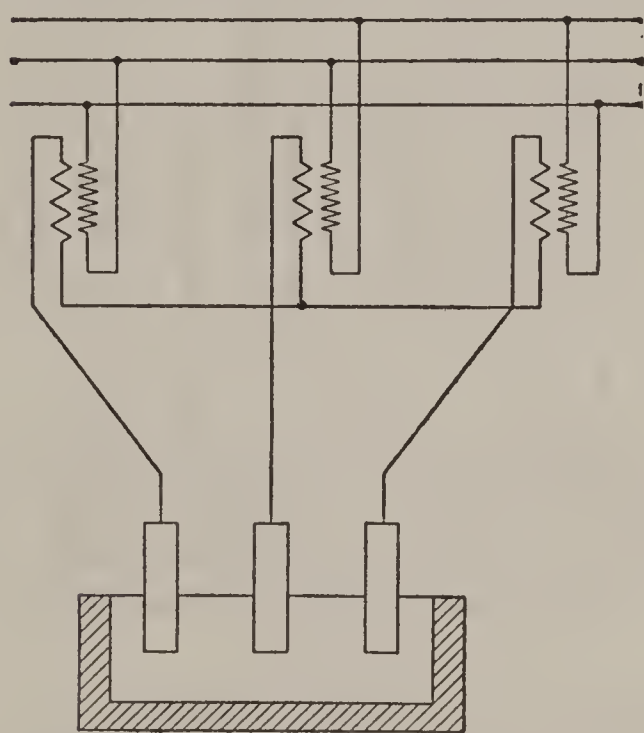


FIGURE 51.—Electrical connections of a three-phase furnace.

450 kilowatts at Kerrouse, in the same district. The company originally manufactured only calcium carbide, but later took up ferro-alloy production on a large scale in conjunction with the production of carbide. Livet is connected with Grenoble, which is on the main line of the Paris, Lyons & Marseille Railroad, by an electric and steam tramway.

POWER SUPPLY.

The company develops its own power on the Romanche River, where a head of 200 feet is utilized by means of a $1\frac{1}{2}$ -mile tunnel. The

cost of power is said to be \$10.66 to \$13.33 per kilowatt-year, or 0.12 to 0.15 cent per kilowatt-hour. At high water there is a maximum of 12,000 to 14,000 kilowatts available, but for the four months from November 1 to March 1 this drops to 7,000 kilowatts, some of the furnaces being shut down and others operated with reduced power.

The first installation of generating machinery, which is still in use, consists of six single-phase dynamos of 900 kilowatts each, delivering current of 55 cycles directly to six furnaces at 55 volts. A more recent installation contains three 2,200-kilowatt three-phase machines, connected in parallel and generating current at 2,300 volts and 25 cycles. This is stepped down by transformers at the furnaces. In addition to the equipment mentioned there are several smaller generators used for general purposes.

GENERAL PLAN OF THE WORKS AND FURNACES.

The plant is arranged so as to require considerable rehandling of materials to be charged into the furnaces. There are 3 furnace houses of stone-and-steel construction, 35 feet wide, 75 feet long, and about 20 feet high. Each building contains three furnaces, which are set up against one wall with the transformers in a room directly outside the wall. Or, if the furnaces are supplied directly by the generators, the arrangement is the same except that there are no transformers. The space in front of the furnaces is used as a tapping floor. Both carbide and ferro-alloy are tapped into small ladle trucks and dumped in an adjoining shed for cooling and packing. Around the top of each furnace there is a steel floor for charging.

There are six Keller furnaces of the type previously described (fig. 40), working on ferro-alloys and three on calcium carbide. The carbide furnaces take from 1,200 to 2,200 kilowatts each, while the ferro-alloy furnaces are from 900 to 1,000 kilowatts capacity, making a total of about 6,000 kilowatts being used in ferro-alloy production. The furnaces directly connected to the single-phase generators receive current at 50 volts at the furnace. The furnaces on the three-phase circuit have a single-phase transformer to each furnace as shown in figure 50, so that the voltage varies from 50 to 70 volts at the furnace.

PRODUCTS.

Three grades of ferrosilicon, containing 25, 50, and 75 per cent silicon, are manufactured at these works. The charge consists of iron turnings, quartzite, and anthracite coal. Only iron turnings containing less than 0.10 per cent phosphorus are used to obtain the iron. The cost of turnings at Livet is about \$14 per ton. The quartzite is ground and contains 94 to 96 per cent SiO_2 , with some phosphorus. Sand is not used to supply the silicon at this plant as it appears to cause irregular operation of the furnaces. For a reducing agent, anthracite coal containing 7 per cent ash, 0.013 per cent phosphorus, and 0.41 per cent sulphur is used. The materials are weighed and thoroughly mixed before being charged. About 500 pounds of 30 per cent alloy is tapped every 2 hours. The 30 per cent ferrosilicon requires 3,500 kilowatt-hours per ton, and the 50 per cent alloy 6,800 kilowatt-hours per ton. After the slag has been tapped into a truck ladle it is skimmed from the metal by means of a green log placed crosswise on a wooden handle. The molds are lined with sand. After having been allowed to cool the ferrosilicon is broken up with a hammer and packed into wooden kegs.

Ferrochrome of all grades is manufactured at Livet. The raw materials used consist of chromite ore containing 35 to 40 per cent chromium, with traces of phosphorus and sulphur, and anthracite coal as a reducing material.

Other ferro-alloys made are silicomanganese, ferrotungsten, ferrotitanium, and ferrovanadium. The total yearly production of the Keller works consists of about 1,500 tons of 25 to 30 per cent ferro-silicon, 3,000 tons of 50 per cent ferrosilicon, 6,000 tons of ferrochrome, 1,500 tons of silicomanganese, 6,000 tons of calcium carbide, and smaller quantities of the higher priced ferro-alloys.

**SOCIÉTÉ ÉLECTRO-MÉTALLURGIQUE PROCÉDÉS PAUL GIROD,
UGINE, SAVOIE, FRANCE.**

In 1898, Girod began the experimental work on the manufacture of ferro-alloys in the electric furnace at Albertville, Savoie, France, with a small 20-kilowatt furnace. In 1899 he opened at Albertville a small works of 38-kilowatt capacity. Later in the same year the erection of a larger plant was begun at that place. The Albertville plant had a capacity of 750 to 950 kilowatts, and consisted of 5 furnaces, each taking 150 to 200 kilowatts. The plant has since been dismantled and the equipment moved to UGINE.

In 1903, the construction of new works was begun at Courtepin, Switzerland. The plant uses 2,250 kilowatts of 25-cycle three-phase current, which is delivered at 32,000 volts and stepped down to 50 to 75 volts at the works. The furnaces at Courtepin are used almost entirely in the production of ferrosilicon containing 25 to 30 per cent, 50 per cent, 75 per cent, and 95 per cent silicon.

In June, 1903, the present company was organized with a capitalization of \$360,000, which has been increased to \$2,400,000, of which \$1,000,000 is in bonds. In 1903 the works at UGINE were erected. In addition to the works mentioned, there is a power plant at St. Gervais, Switzerland.

UGINE is situated on a branch line of the Paris, Lyon & Marseille Railway between Annecy and Albertville. The works are about three-quarters of a mile from the main line, with which connection is made by an electric tramway for freight haulage, which is operated by the Girod Co. The buildings are all on the north bank of the Arly River.

POWER SUPPLY.

The total capacity of all of the Girod power plants is about 28,000 kilowatts. There is a possible development of 30,000 kilowatts more, a total of 58,000 kilowatts. The average cost of power from all sources is about \$18.66 per kilowatt-year, or 0.21 cent per kilowatt-hour.

A short distance up the Arly River from the plant is the dam of one of the power plants of the Girod Co. The water is led directly to the plant by a pipe line, the total fall being 416 feet. Ten dynamos, driven by 10 Pelton wheels, generate 6,000 to 7,000 kilowatts of single-phase current, which is conducted directly to the furnaces in the old furnace house.

At La Fayette, also on the Arly River, at a considerable distance from Ugine, 9,000 to 15,000 kilowatts of three-phase current is developed. This is transformed at the works from 45,000 to 50 or 75 volts for the furnaces and 2,500 volts for conversion to direct current. About 4,500 kilowatts of three-phase current is obtained from a plant at St. Gervais, Switzerland, on the Bonnart River. To supply a deficiency during the low-water months, from October 1 to April 1, 6,000 to 7,500 kilowatts, all three-phase at 4,500 volts, is leased from another electrometallurgical company at Veuthin.

The present capacity of the two Girod power plants on the Arly River that supply the Ugine plant is about 24,000 kilowatts. For

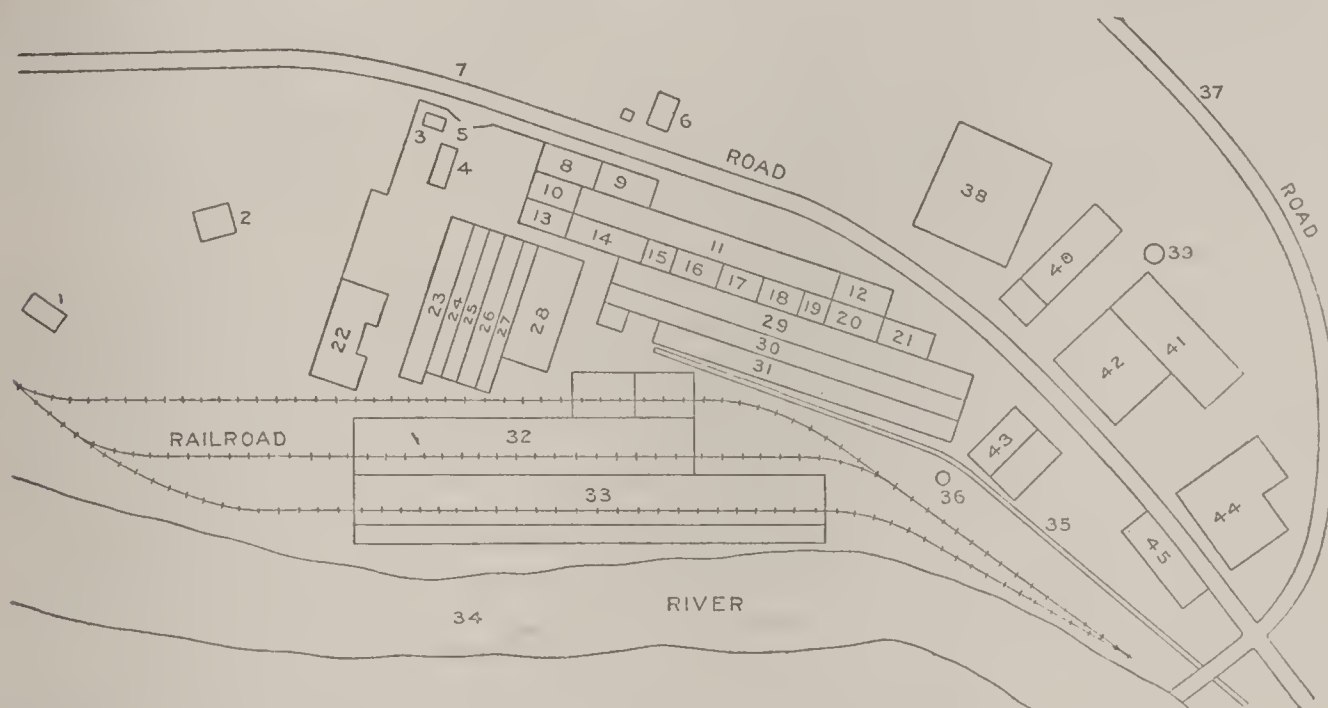


FIGURE 52.—Plan of Girod ferro-alloy plant: 1, garage; 2, villa; 3, porter's lodge; 4, dwelling house; 5, portal; 6, dwelling house; 7, road; 8, fitting and tinsmith shop, 75 by 25 feet; 9, metal-sorting shop, 75 by 25 feet; 10, laborers' dining hall; 11, raw-materials storeroom, 285 by 40 feet; 12, workshop, 60 by 40 feet; 13, dining room; 14, lavatories; 15, foremen's office; 16, packing shop, 60 by 40 feet; 17, hammer with falling weight of 220 pounds; 18, foremen's office; 19, shop for fitting up electrodes; 20, repairing shop; 21, workshop for preparation of graphite; 22, general office; 23, transformer house, 175 by 25 feet; 24, furnace house, 160 by 25 feet; 25, covered yard, 160 by 25 feet; 26, furnace house, 160 by 25 feet; 27, transformer house, 160 by 25 feet; 28, ferro-alloys store house, 130 by 50 feet; 29, covered yard, 440 by 30 feet; 30, old furnace house, 440 by 30 feet; 31, power house, 360 by 40 feet; 32, stock shed, 360 by 50 feet; 33, raw-materials storehouse; new smelting works, 520 by 75 feet; 34, Arly River; 35, conduit pipe; 36, chimney, 125 feet high; 37, road to Ugine; 38, graphite and carbon storehouse, 150 by 100 feet; 39, chimney; 40, electrode storehouse, 170 by 40 feet; 41, furnace house for electrode manufacture, 170 by 40 feet; 42, electrode presses, etc., 110 by 75 feet; 43, sawmill; 44, graphite storehouse; 45, dwelling house.

four months, from October to February, and sometimes to April, the current delivered is reduced to about 12,000 kilowatts when leased power is used. All of the power developed by the Girod plants, except about 6,000 kilowatts that supplies the electric steel works, is used in ferro-alloy manufacture.

GENERAL PLAN OF THE WORKS.

The works cover about 500,000 square feet, including the electrode department, which is operated in conjunction with the ferro-alloy plant. The general layout of the plant is shown in figure 52.

The old furnace building is of stone construction, 440 feet long, 30 feet wide, and 18 feet high. The power house runs parallel to it,

with a wall between the furnaces and the generators, so that a furnace is not more than 20 feet from its generator. The ends of the furnace building are open, with an open ventilation on the top for the length of the building. Adjoining the furnace building on the side away from the furnaces is an open shed into which the finished products are dumped to cool. Ore, sand, and coal are weighed and mixed on the floor, from which they are shoveled into the furnace tops.

The furnaces in the old works are chiefly used for the manufacture of the higher-priced alloys. There are 18 furnaces, ranging in size from 150 to 560 kilowatts capacity each. Both crucible and electrode furnaces are installed. The furnaces, about 6 feet apart, are set along one wall, with the wheel for hand regulation of electrodes, as well as the electrical instruments, on the opposite wall. The electrodes are suspended by cables running over blocks on the steel trusses of the roof.

At the new works the raw materials are unloaded from the railway cars into the open raw-material shed. An electromagnet crane is used in unloading iron turnings. The ores are then carried in small electrically hauled cars to the new furnace house, where they are elevated to bins above the furnaces. Weighing, mixing, and charging are done here from a floor at the level of the furnace top, considerable labor being thus saved over that necessary at the old plant. The transformers for the three-phase current are set in rooms parallel to each furnace house. The furnaces are set close to the dividing wall. Beyond the other wall of the furnace house is an open shed, into which the hot products are dumped. The furnaces in the new works are all of the electrode type, taking 375 to 900 kilowatts. The electrodes are supported by cables running to the steel framework over the furnaces. Regulation is by hand, as in most ferro-alloy furnaces.

PRODUCTION.

The Girod Co. employs 400 to 500 men in the production of ferro-alloys, the yearly value of which is between \$2,000,000 and \$3,000,000. The quantity produced varies considerably from year to year, but the production is estimated at 15,000 to 20,000 tons; ferrosilicon, 30 to 90 per cent silicon, 6,000 tons; ferrochrome, 5,000 tons; ferrotungsten, 1,000 tons; silicomanganese, 2,000 tons; other alloys, 1,000 tons. The details of manufacture of these alloys is about as is later described in general under each ferro-alloy.

MERAKER ELECTRIC SMELTING CO.

The works of the Meraker Electric Smelting Co. are at Kopperaaen, Norway, 55 miles east of Trondjhem on the main line of the railroad to Stockholm. The company operates an electric-furnace plant for

the manufacture of calcium carbide and ferro-alloys at Kopperaaen, and a pulp-grinding plant about 3 miles west, at Meraker. The concern has been manufacturing since 1905, and, in spite of several fires that almost destroyed the plant, has been very successful. In the summer of 1912, ferrochrome was the only alloy manufactured, but it was the intention of the company to begin the production of other ferro-alloys, especially ferrotitanium, during the next year.

POWER SUPPLY.

The company owns its own power plants. At Kopperaaen there is a fall of 250 feet in the Kopperaaen River. The river is fed by two lakes regulated so as to give a minimum flow of 160 cubic feet per second and a maximum flow of 210 cubic feet per second. At present about 2,250 kilowatts is developed at the foot of this fall at Kopperaaen, and 750 kilowatts more is received from the power plant at the pulp mill in Meraker. The total power used in the electric-furnace plant is about 3,000 kilowatts. At Kopperaaen there are two 1,125-kilowatt direct-current generators delivering current at the furnaces at 65 volts and about 14,250 amperes. There are also several small units for power and lighting purposes. Of the 3,000 kilowatts now being used half is for ferrochrome manufacture and the rest for calcium carbide manufacture. The cost of power is stated to be \$8.30 per kilowatt-year, or 0.095 cent per kilowatt-hour. At a point farther up the river 3,000 kilowatts more is to be developed this year, making a total of 6,000 kilowatts used in the electric furnaces. When desired it is possible to develop 4,000 kilowatts more at this point.

GENERAL PLAN OF WORKS AND FURNACES.

Raw materials are unloaded into bins near the Kopperaaen railroad station, about one-half mile from and 300 feet above the plant. There are bins for coal, limestone, and chromite. Besides the bins there is a shipping house where boxes for ferrochrome and iron drums for calcium carbide are made. An aerial tramway runs from the bins and shipping house to the main storage bins about 200 feet from the furnace house but at an elevation 200 feet higher. Here the raw materials are dumped from the tram buckets, and boxes and drums of product are loaded into the buckets after having been hauled up the hill from the furnacehouse by cable skips. At the level there is a kiln for calcination of the limestone. From the bins 200 feet above the furnace house the ores, coal, and lime are lowered to the furnaces by an incline skip discharging into bins above and outside the furnace house.

From these bins the materials are drawn into cars running around a floor in the furnace room about 15 feet above the main working floor, and are dumped into small chutes at each furnace. The chutes,

two to each furnace, are about 4 feet deep, and $1\frac{1}{2}$ feet by 1 foot in cross section. From these chutes the ore, coal, and lime are drawn into small cars and dumped into the furnaces about 15 feet away.

The furnace house is about 110 feet long, 75 feet wide, and 22 feet high, and is built of structural steel with brick-and-stone walls. An open ventilator in the roof runs the length of the building. The gases from all the furnaces, except one that is covered and connected with a stack, escape into the furnace room and pass out through the roof.

The main floor of the furnace house is cement, with tracks inlaid for the hauling of small ladle trucks. There is an open space at one end for the dumping of hot carbide and ferrochrome. Around each furnace at the level of the furnace top there is a steel floor, supported by iron pillars. The furnaces are charged from these floors, which are about 15 by 20 feet.

There is a steel floor 10 feet wide suspended above the charging floor. The main bins are discharged into the individual furnace chutes from this floor. This floor also extends out over each furnace to make charging of the electrodes less difficult.

There are five 750-kilowatt furnaces of the Alby type (fig. 44), arranged two on each side and one at the end of the room. Only four of these are operated at a time, one being kept in reserve. There is also a 450-kilowatt furnace of this arc type with a shallow hearth for refining. Three of these furnaces are operated with direct current at 60 volts and one by single-phase current at the same voltage. The electrode holders are of the type shown in figure 45. A spare one is left to facilitate charging at each furnace. Jointed connections for continuous feeding are not used. In the four open-top furnaces an electrode 48 by 10 inches by 6 feet long lasts about 10 days, whereas in the covered-top furnace they last 14 to 16 days. The electrodes project about 10 to 12 inches into the charge, which is not molten at the top, and are consumed down to about 2-foot to 1-foot lengths.

The lining of magnesite and tar in one ferro-alloy furnace had been in use for 22 months and the other 6 months. The power had not been off these furnaces at all during that time except for about 15 minutes a week when electrodes were changed. The current is conducted from the power house at the end of the furnace room by copper cables. The connection to the carbon-block base of each furnace is made by many small wires leading to the main cable line.

PRODUCTS.

The Meraker company produces 4,000 tons of calcium carbide and 2,000 tons of ferrochrome annually. Turkish ore and English anthracite coal are used for the manufacture of the ferrochrome. In calcium-carbide production the power consumption is about 0.375 kilowatt-year per metric ton or 1.5 kilowatt-hours per pound. The power

consumption for ferrochrome is 0.75 kilowatt-year per metric ton or 3 kilowatt-hours per pound. The carbide and ferrochrome are tapped every 2 and 4 hours, respectively. The carbide is sized before being packed into 50-kilogram (110-pound) drums, and the ferrochrome is broken into lumps less than 6 inches square and packed into 200-kilogram (440-pound) boxes.

This company makes only ferrochrome that can be produced without the refining process described later, so that the carbon content of its product varies from 2 to 10 per cent. It is very easy to make a ferrochrome of 5 per cent carbon content without refining the product. The cost of manufacture of carbide is given at about \$30 per metric ton. All products are sold through a syndicate of manufacturers organized to prevent overproduction if possible. The company employs 120 men, part of whom work an 8-hour shift and part 10 hours. The men in the furnace house are on an 8-hour shift. The minimum wage is \$1.25 for a 10-hour shift.

FERRO-ALUMINUM.

Aluminum is usually used by the steel maker in the form of metallic aluminum, but sometimes a ferro-aluminum is made. Metallic aluminum is supplied in half-round sticks or notched ingots. Ferro-aluminum is made containing 5, 10, and 20 per cent aluminum, with the rest iron. The other iron-aluminum alloys, such as ferro-aluminum silicon, are discussed later.

The manufacture of ferro-aluminum consists simply in melting aluminum and iron together in the desired percentages. This may be done in either an electric furnace or a combustion furnace of the crucible type. Aluminum or ferro-aluminum is added in the ladle and serves only for the deoxidizing of the steel.

Iron containing 5 to 6 per cent aluminum is fragile.^a At 7 per cent aluminum the alloy has a crystalline structure. An iron aluminum alloy containing over 20 per cent aluminum has no magnetism. Ferro-aluminum containing from 0 to 34 per cent aluminum consists of mixed crystal of iron and aluminum.^b When 34 to 52 per cent aluminum is present there are crystals, and with 52 to 100 per cent aluminum the alloys FeAl_3 and Al are found together.

FERROBORON.

Ferroboron is not as yet very commonly used, as the effect of boron on steel has not been extensively investigated. Ferroboron is manufactured directly by the reduction of colemanite $\text{Ca}_2\text{B}_6\text{O}_{11} \cdot 5\text{H}_2\text{O}$ and iron with carbon in an electric furnace of one of the types already

^a Guillet, L., *Étude Industriel des Alliages Métallique*, p. 930.

^b Robin, F., *Traité de Métallographie*, 1912, p. 304.

described. The analysis of a typical Pacific Coast colemanite is as follows:^a

	Per cent.
B ₂ O ₃	27. 23
CaO.....	28. 80
H ₂ O.....	25. 30
Fe ₂ O ₃	Trace.
Insoluble.....	0. 74

In the western United States there are large deposits of this mineral, the principal use for which is in the manufacture of borax. In manufacturing ferroboron, carbon is mixed with the colemanite and iron turnings or iron ore in sufficient quantity to produce as complete reduction as possible, although the slags are usually high in boron. The product of the reduction with carbon in the electric furnace is of various grades containing 10 to 12, 20 to 25, and 30 to 35 per cent boron. A typical ferroboron of the latter class had the following composition:

	Per cent.
Boron.....	32. 75
Carbon.....	2. 875
Sulphur.....	. 03
Phosphorus.....	. 005

In a French patent Girod claims that the presence of boron either as colemanite or as the alloy in a charge gives a more fluid slag in the manufacture of other ferro-alloys, and if it goes into the alloy increases its value.

Hansen^b has patented a method for the production of ferroboron from colemanite with ferrosilicon as a reducing agent. In this way there is no carbon in the ferroboron, but it is apt to contain considerable silicon. The colemanite is powdered and briquetted with enough ferrosilicon to provide silicon for complete reduction of the boron and to slag off the calcium oxide. Water glass or some other suitable material is used as a binder. The briquets are subjected to a temperature of over 2,000° C., giving ferroboron and a calcium silicate slag.

Ferroboron containing about 10 per cent boron seems to give the best results. Ferroboron is added in the ladle and may be used for deoxidizing, desulphurizing, and for a fixed addition of boron to steel.

Guillet^c states that the best commercial boron steel is that containing 0.22 per cent carbon and 0.5 per cent boron. Boron steels are practically useless unless tempered, when they have high tensile strength and elastic limit and great resistance to shock.

^a Iwai, K., and Ballagh, J. L., Investigation of ferroboron: Min. and Sci. Press, vol. 99, 1909, p. 185.

^b U. S. Patent 982135, Jan. 17, 1911.

^c Guillet, L., Étude Industriel des Alliages Métallique, p. 372.

FERROCHROME.**HISTORY.**

More ferrochrome is manufactured in the electric furnace than any other ferro-alloy except ferrosilicon. The alloy of iron and chromium, called ferrochromium or ferrochrome, has been known since 1820 through the studies of Faraday and Stodarts, and also by the work of Berthier. Berthier found that by adding ferrochrome containing 17 per cent chromium to steel, so as to give $1\frac{1}{2}$ per cent chromium in the steel, a product of better quality was obtained. Baur found 40 years later that ferrochrome was the best agency to use for adding chromium to steel. Other experimental work on chrome alloys was done by Fremy, Percy, and Mushet.

In 1873 Biermann, of Hanover, announced that his firm was producing ferrochrome commercially. In 1877 the English firm of Hadfield put on the market some ferrochrome containing 10 to 30 per cent chromium. The French firm of Holtzer & Co. also began manufacture. By 1886, as the value of chromium in steel became more understood, there were several regular producers of ferrochrome. In 1899 there began the production of ferrochrome in the electric furnace. Up to that date all of it had been made in either the crucible or the blast furnace.

WORK OF EXPERIMENTERS.**NEUMANN.**

Neumann^a investigated the use of ferrosilicon as a reducing agent for chromite ($\text{FeO} \cdot \text{Cr}_2\text{O}_3$) in the electric furnace in an effort to produce a carbon-free alloy. The experiments were performed in an electric furnace holding about 1 kilogram of charge and operating with 100 amperes and 30 to 50 volts. Ferrosilicon containing 91.65 per cent silicon and 1.03 per cent carbon was used as a reducing agent. The chromite contained 31 per cent chromium and 13 per cent iron. A mixture of ($150\text{Al}_2\text{O}_3 + 100\text{CaO}$ or $100\text{Al}_2\text{O}_3 + 100\text{CaO}$) was placed in the furnace, 250 grams being used. When this was fluid, 700 grams of a mixture of chrome ore and ferrosilicon was added. About 140 grams of metal was obtained, which contained 38.05 per cent chromium, 3.09 per cent silicon, and 1.56 per cent carbon. In another experiment the metal contained 43.16 per cent chromium, 2.53 per cent silicon, and 0.87 per cent carbon. There seems to have been a heavy loss through slagging of chromium. From these experiments it is evident that the resulting alloy from reduction with silicon will be high in silicon and also contain considerable carbon if there is any present in the silicon used.

^a Neumann, G., Herstellung von Ferro-liegierung im elektrischen Ofen: Stahl und Eisen, vol. 28, 1908, p. 356.

MOISSAN.

Moissan ^a reduced chromite in the electric furnace with carbon as a reducing agent and obtained a ferrochrome that contained 60.9 per cent chromium, 31.6 per cent iron, 6.1 per cent carbon, and 1.1 per cent silicon. He also conducted experiments on decarburization of ferrochrome. A ferrochrome containing 61.81 per cent chromium, 30.02 per cent iron, 7.53 per cent carbon, and 0.33 per cent slag was broken into pieces and fused under a bath of liquid lime. A fine ground metal was obtained, which contained 64 per cent chromium, 35.12 per cent iron, 0.70 per cent carbon, and 10.22 per cent slag.

EXPERIMENTAL SMELTING OF CHROMITE IN THE ELECTRIC FURNACE.

CONSTRUCTION OF FURNACE USED.

Some experiments were made by the writer ^b in 1912 on the production of ferrochrome from chromite with carbon as a reducing agent. The experiments were performed in a furnace lined with magnesite, having one upper graphite electrode and a conducting hearth of iron rods embedded in magnesite. The interior dimensions of the crucible were: Width 4 inches, length 8 inches, and depth 8 inches. The furnace took 8 to 12 kilowatts of single-phase current at 30 to 50 volts. The furnace held 12 to 15 pounds of the cold charge.

MATERIALS USED IN CHARGE.

The chromite used was a foreign grade used commonly for furnace linings. Analyses of the chromite, coke, lime, and fluorspar used are given below:

Analyses of raw materials used in making ferrochrome.

Chromite.		Coke.		Lime.		Fluorspar.	
Constituent.	Per cent.	Constituent.	Per cent.	Constituent.	Per cent.	Constituent.	Per cent.
Cr ₂ O ₃	¹ 46.35	Fixed carbon..	81.48	CaO.....	92.76	CaF ₂	99.00
FeO	² 21.45	Volatile com-		SiO ₂	1.34	CaCO ₃80
SiO ₂	5.48	bustible		Al ₂ O ₃ -Fe ₂ O ₃ ..	1.90	Al ₂ O ₃	8.50
Al ₂ O ₃	12.80	matter.....	.25	MgO.....	Trace.		
MgO	10.08	Ash.....	17.92	P.....	.06		
CaO	Trace.	Moisture.....	.35				
P013	FeO.....	1.20				
S45	SiO ₂	13.62				
		CaO.....	.85				
		P.....	.08				
		S.....	.54				

¹ 31.7 per cent Cr.

² 16.6 per cent Fe.

^a Moissan, H., The electric furnace, p. 152. Translated by V. Lenher, 1904.

^b Keeney, R. M., The production of steels and ferro-alloys directly from ore in the electric furnace: Iron and Steel Inst. (Carnegie Scholarship Memoirs), vol. 4, 1912, p. 108.

In these experiments, contrary to the usual European practice in ferrochrome manufacture, lime was used as a flux to show its desulphurizing and dephosphorizing effect with an ore not so pure as the usual ore used in ferrochrome production. In usual practice only the most pure raw materials available are charged, so as to avoid the excess slag caused by adding lime. The only slag formed is from the coal or coke and chromite.

RESULTS OBTAINED.

Five experiments were made in which the furnace was charged and tapped at the completion of reduction. In these experiments the chromium percentage of the alloy seemed to run considerably lower than had been calculated, although this result is explained in a measure by the fact that the carbon in the metal was not considered in calculating the theoretical percentage. The carbon content was high in all of the alloys and does not seem to be influenced by the carbon in the charge as long as there is excess, which was the case. The phosphorus content also was high in all of the products. Seemingly all of the phosphorus charged was concentrated in the metal. This was caused by the excessive reducing conditions which did not permit the slagging of the phosphorus as calcium phosphate. The ore contained considerable sulphur—0.45 per cent—but almost all of it passed into the slag, probably as calcium sulphide. There was considerable loss of chromium and iron in the slag. In some cases the percentage of iron in the slag exceeded that of the chromium, but in no case did the percentage of chromic oxide in the slag exceed 9 per cent, or the percentage of chromium, 4.69 per cent, showing that in the endeavor to keep the carbon in the alloy low the proportion of coke added was not sufficiently in excess for complete reduction or that the furnace was not hot enough. The former was probably the case, as the furnace was operated as an arc furnace throughout the experiments.

In two experiments (Nos. 6 and 7 of table following) the furnace was left full and several tappings were made during the run, the usual practice of ferrochrome manufacture thus being followed. Commercial ferrochrome is usually 5 to 10 per cent carbon, containing 60 to 65 per cent chromium.

Details of the seven experiments mentioned above are presented in the following table:

Details of experiments in the production of ferrochrome directly from ore.

Experiment No.	1	2	3	4	5	6			7		
						a	b	c	a	b	c
Charge:											
Chromite.....	10.00	10.00	10.00	10.00	10.00	50.69	68.20	58.90	62.60	72.70	68.00
Coke.....	1.87	1.87	1.87	1.63	1.36	6.50	9.31	8.72	5.28	8.32	5.43
Lime.....	1.25	1.25	1.25	1.25	1.25	.67	.60	1.23	.79	.56	.94
Fluorspar.....	.12	.12	.12	.12	.12	.106	.099	.108	.121	.119	.104
Constituents of alloy product:						.071	.105	.055	.020	.023	.027
Chromium.....	51.80	48.60	49.70	52.70	57.10	9.15	12.60	10.72	8.78	11.40	11.35
Carbon.....	10.05	8.74	11.50	3.72	9.28	32.60	34.20	29.20	36.84	30.50	29.40
Silicon.....	.81	1.10	1.26	.87	.35	6.10	14.82	15.43	8.73	16.99	18.10
Phosphorus.....	.128	.133	.110	.106	.120	65.50		65.50		65.50	
Sulphur.....	.073	.038	.096	.061	.085	.312		.312		.418	
Slag, Cr ₂ O ₃	8.03	7.57	6.96	8.98	8.81	2.66	2.13	3.21	1.50	2.51	7.00
Slag, SiO ₂	23.40	24.60	23.40	20.25	26.60	2.66	2.13	3.21	1.50	2.51	7.00
Slag, FeO.....	9.02	7.29	18.96	15.10	10.17	2.66	2.13	3.21	1.50	2.51	7.00
Calculated percentage of chromium.....	65.50	65.50	65.50	65.50	65.50	2.66	2.13	3.21	1.50	2.51	7.00
Carbon per pound of iron and chromium.....	.312	.312	.312	.271	.228	2.66	2.13	3.21	1.50	2.51	7.00
Calculated weight.....	4.45	4.00	4.47	3.87	5.06	2.66	2.13	3.21	1.50	2.51	7.00
Ferrochrome tapped.....	3.5	4.87	5.15	2.44	2.66	2.66	2.13	3.21	1.50	2.51	7.00
Do.....	78.5	89.9	86.3	62.95	53.30	2.66	2.13	3.21	1.50	2.51	7.00
Electrode consumption per ton tapped.....	210.00	150.00	142.00	204.00	189.00	2.66	2.13	3.21	1.50	2.51	7.00
Power consumption per pound of alloy tapped, kilowatt-hours.....	3.18	2.73	2.57	6.38	4.71	2.66	2.13	3.21	1.50	2.51	7.00
Power consumption per ton of alloy tapped, kilowatt-years.....	.73	.62	.59	1.46	1.08	2.66	2.13	3.21	1.50	2.51	7.00

a Total, 17.92 pounds.

b 11.01 pounds.

In these experiments the percentage of chromium in the alloy closely approached the theoretical. The percentage of carbon was also lower in these alloys because of the decarburizing effect of the unreduced oxide in the charge above the molten metal. The slag loss of iron and chromium was higher in these runs than in the intermittent experiments because of unreduced oxide getting into the slag on tapping.

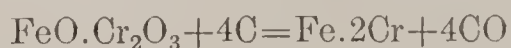
A total of 50 pounds of ferrochrome was tapped during the experiments, or an average extraction of 69.5 per cent. The average iron content of the slag for all experiments was 13.4 per cent FeO (10.4 per cent Fe); the average chromium content of the slag was 10.3 per cent Cr_2O_3 (6.6 per cent Cr). In the intermittent experiments the average percentages in the slag were 12.1 per cent FeO (9.4 per cent Fe) and 8.07 per cent Cr_2O_3 (5.32 per cent Cr). Thus, of the loss of 30.5 per cent of material that was not tapped, 11.9 per cent was chromium and 18.6 per cent was iron, on the assumption that of the total average loss 39 per cent was chromium and 61 per cent iron. A large part of the ferrochrome not tapped stuck in the furnace. Although the average electrode consumption was 158 pounds per ton, the figure obtained in experiment No. 6, 48.1 pounds, would probably be attained in practical operation. The average power consumption was 3.7 kilowatt-hours per pound, or 0.85 kilowatt-year per ton.

The conclusions drawn are, first, ferrochrome can be easily manufactured directly from chromite in the electric furnace; second, the percentage of carbon in ferrochrome can not be kept low by regulating the carbon charged without excessive loss of chromium in the slag; third, the percentage of carbon in the ferrochrome must be regulated by decarburization with an oxide slag of iron or chromite after the slag from reduction has been tapped off; fourth, the proportions of silicon and phosphorus can not be kept low in the alloy under the strong reducing conditions necessary; fifth, sulphur can be easily slagged; sixth, as the addition of lime does not seem to aid in reducing phosphorus, it is advisable to use no lime but raw materials as pure as possible; and, seventh, the power consumption should not exceed 3.7 kilowatt-hours per pound of ferrochrome tapped or 0.85 kilowatt-year per ton.

THEORY OF CHROMITE SMELTING.

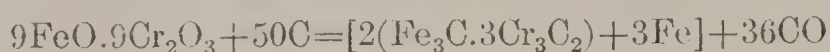
REDUCTION WITH CARBON.

In the reduction of chromium from chromite to form ferrochrome with the iron reduced, reduction of chromic acid begins at $1,185^\circ \text{C}.$ ^a Reduction takes place according to the reaction:



^a Greenwood, H. C., Slade, R. E., and Pring, J. N., Reduction of refractory oxides, production of ferroalloys, and formation of carbides: Trans. Chem. Soc. (England), vol. 93, 1908, p. 1484; Electrochem. and Met. Ind., vol. 7, 1909, p. 119.

A pure chromite ore contains 68 per cent Cr_2O_3 and 32 per cent FeO . For reduction of pure chromite 30 parts of carbon are theoretically necessary for every 100 parts of iron and chromium reduced. In practice this amount is exceeded considerably. In ferrochrome manufacture practically all of the reduction is performed by the solid carbon and not by carbon-monoxide gas. On the basis that the double carbide $\text{Fe}_3\text{C} \cdot 3\text{Cr}_3\text{C}_2$ forms in reduction with carbon as has been found to be the case, the reduction reaction is as follows:



The ferrochrome resulting would contain theoretically 10.4 per cent carbon, 31.4 per cent iron, and 58.2 per cent chromium. For reduction of 100 parts of this alloy 37 parts of carbon are necessary, the above reaction explaining the difference between the theoretical and the practical proportion of carbon necessary, if calculation is made on the assumption that the double carbide is not formed.

Carnot and Goutal isolated $\text{Fe}_3\text{C} \cdot 3\text{Cr}_3\text{C}_2$ from ferrochrome containing 59 per cent chromium and 9 per cent carbon.^a They found $3\text{Fe}_3\text{C} \cdot \text{Cr}_3\text{C}_2$ in chrome steel containing 2 per cent chromium and 2 to 0.5 per cent carbon. In the electric furnace Moissan obtained Cr_3C_2 . By treating 50 per cent ferrochrome with hydrochloric acid Behren and Van Linge obtained Cr_3FeC_2 which corresponds closely to the double carbide isolated by Carnot and Goutal. Williams^b, using the electric furnace, obtained $3\text{Fe}_3\text{C} \cdot 2\text{Cr}_3\text{C}_2$. All of these researchers show the existence of double carbides of varying percentages but all consisting of Fe_3C and Cr_3C_2 .

The characteristics of ferrochrome of different percentages of carbon are widely variable. All of the alloys described contain 60 per cent or more of chromium. Ferrochrome containing 0.5 per cent carbon is bright lead gray in color, with the structure parallel to the bed of cooling. It breaks with difficulty, has a high compressive strength, and is one and one-half times as magnetic as soft steel of about 0.2 per cent carbon.

Ferrochrome containing 1.8 per cent carbon is gray in color, and breaks into thin plates many of which show faces of one-third millimeter perpendicular to the bed of cooling. Naturally, it is a little magnetic, but on magnetizing it has more than twice the magnetism of soft steel.

The 2.5 per cent carbon ferrochrome has the gray color of polished aluminum. It is weak and is about as magnetic as soft steel. With 3.6 per cent carbon, the alloy has a bright-gray color. It has little brittleness and has magnetism equal to ferrochrome containing 1.8 per cent carbon. Ferrochrome containing 4.5 per cent carbon is

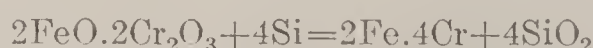
^a Guillet, L., *Étude Industriel des Alliages Métallique*, p. 419.

^b Robin, F., *Traite des Métallographie*, 1912, p. 301.

dull gray, has little brittleness, and has magnetism equal to that of soft steel. Ferrochrome containing 5.5 per cent carbon has the gray color of bright iron. It breaks up a little, has a lower compressive strength, and a magnetism equal to that of 5.5 per cent carbon. With 7.5 and 9 per cent carbon the ferrochrome is gray, contains tin-gray oriented needles, breaks up, has no magnetism, and is very porous.

REDUCTION WITH SILICON AND OTHER REDUCING AGENTS.

Several United States patents have been issued for the reduction of metallic oxides by the use of silicon as a reducing agent. The reaction is as follows:

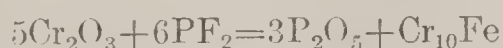


Thus for the reduction of 100 parts of ferrochrome containing 65 per cent chromium, 35 parts of silicon are necessary. In United States patents granted to F. M. Becket ^a based on this reaction it is claimed that a ferrochrome with a silicon content as low as 0.01 per cent or less can be made by this method, although this possibility is not shown in the researches of Neumann with the silicon percentage of the alloy high. Patents based on about the same reaction, with ferrosilicon as a reducing agent have been granted to E. F. Price.^b These processes are expensive, the silicon content in the alloy tends to be high, and as a result they are not widely used in actual manufacture.

Keigelgen and Seward^c reduce chromium oxide with calcium phosphide according to the reaction

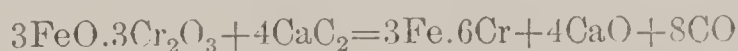


With phosphide of iron they utilize the reaction



The method is not commercially applied.

Becket^d proposes the use of calcium carbide as a reducing agent. He claims that the calcium reduces the metals and that the carbon is oxidized or remains with the resulting alloy, according to the reaction



The method has not been used much commercially because of the expense. For the reduction of 100 parts of ferrochrome containing 65 per cent chromium, 53 parts of calcium carbide are necessary.

^a 854018, May 21, 1907; 866561, Sept. 17, 1907; 891898, June 30, 1908.

^b 865609, Sept. 10, 1907; 852347, Apr. 30, 1907; 862996, Aug. 13, 1907.

^c U. S. Patent 878966, Feb. 11, 1908.

^d U. S. Patent 898173, Sept. 8, 1908.

Rossi,^a using aluminum as a reducing agent in the electric furnace, has produced ferrochrome containing 68.24 per cent chromium, 1.85 per cent silicon, 1 per cent carbon, and 0.5 per cent aluminum from chromite containing 50.29 per cent Cr_2O_3 , 16.01 per cent FeO , 10.72 per cent Al_2O_3 , 4.62 per cent SiO_2 , 16.61 per cent MgO , 0.01 per cent sulphur, and 1.15 per cent CaO .

REFINING OF FERROCHROME.

Decarburization of high-carbon ferrochrome is accomplished by melting the alloy with a slag of lime, chromite ore, and a little fluor-spar, according to the following reaction:



For each 100 parts of completely decarburized alloy 39 parts of pure chromite ore are necessary, or for each 100 parts of ferrochrome containing 10.6 per cent carbon, 31.3 per cent iron, and 58.1 per cent chromium, 36 parts of pure chromite ore are necessary for complete decarburization. The resulting alloy would contain 66.6 per cent chromium and 33.4 per cent iron. In practice the lime is added simply to flux any silica in the ore, to assist in elimination of silicon in the ferrochrome, and to cause possible removal of carbon as calcium carbide, as the operation is performed in an arc furnace. The fluor-spar makes the slag more fluid. Gin^b claims that high temperature is not necessary for decarburizing, but his contention is not sustained in practice.

E. F. Price^c proposes to decarburize ferrochrome by passing an oxidizing gas through the molten metal, but the process is not used in practice.

F. M. Becket^d proposes to make a ferrochrome high in silicon, containing approximately 51.3 per cent chromium, 17.5 per cent iron, 30 per cent silicon, and 1.2 per cent carbon in a first stage. Then he proposes to remove the silicon by adding chromite in the proportion of 6 parts of chromite to 1 part of silicon in the alloy, this proportion of chromite being slightly in excess of that theoretically necessary to oxidize all the silicon to silica. He gets an alloy containing 70 per cent chromium, 28.9 per cent iron, 1 per cent carbon, and 0.10 per cent silicon. The necessity of such a refining process shows the weakness of methods using ferrosilicon or silicon as a reducing agent in that the percentage of silicon in the alloy can not be kept low in one operation without high losses of chromium in the slag.

^a Rossi, A. J., *Ferro-alloys: Min. Ind.*, vol. 12, 1903, p. 693.

^b Gin, G., *Decarburization of ferro-alloys: Trans. Electrochem. Soc.*, vol. 15, 1909, p. 225.

^c U. S. patent 886858, May 5, 1908.

^d U. S. patent 891898, June 30, 1908.

MANUFACTURE OF FERROCHROME.

Until the introduction of the electric furnace in 1899 ferrochrome was manufactured entirely in either the crucible or the blast furnace. The crucible was used only for the production of the higher grades, chromite being reduced with charcoal, and a flux of lime, borax, fluorspar, or water glass being used. In the blast furnace chromite is reduced with some difficulty, with the high fuel consumption of about 3 tons of coal or coke to 1 ton of ferrochrome produced. Theoretically, it should be possible to get a ferrochrome containing 65 per cent chromium from ore containing 60 per cent Cr_2O_3 and 20 per cent FeO , but generally a ferrochrome containing 30 to 40 per cent chromium and 6 to 12 per cent carbon is the product in the blast furnace. Little ferrochrome is now made by either the crucible or the blast furnace, because it can be made cheaper in the electric furnace in localities where electric power is cheap. A small amount of carbon-free ferrochrome is now made by the thermit process.

ORES AND RAW MATERIALS USED.

The greater part of the ores now used in the electric furnace manufacture of ferrochrome come from Turkey, New Caledonia, Cuba, and New South Wales. There are considerable deposits of chromite ore in Canada and California, but they are so far from the market that the production does not amount to much. The analyses of some of the ores are given below:

Analyses of chromite ores.

Component.	Source of ore.			
	Turkey.	New Caledonia.	California.	Cuba.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Cr_2O_3	51.7	55.7	52.68	50.0
FeO	14.2	13.2	15.30	18.57
Al_2O_3	14.1	16.2	11.49	12.44
MgO	14.3	9.8	16.23	13.38
SiO_2	3.5	.2	3.40	3.82
CaO	1.7	.25		2.16
H_2O3	1.05	.94	
MnO2	.15	
P_2O_505		.45
S.....				.69

The foreign manufacturers use only chromite containing less than 0.1 per cent phosphorus and sulphur and about 50 per cent Cr_2O_3 . They use no lime in the charge, as it has been found preferable by them to use pure materials and keep the amount of slag as low as possible. In most plants high-grade anthracite coal is used as the reducing agent. The only American manufacturers of ferrochrome,

according to their patent specifications, use a lime slag. They sometimes use Cuban ore, which contains considerable phosphorus and sulphur. Coke or charcoal may be used instead of anthracite coal, but the latter is better, as it may be obtained in a more finely pulverized state, a form in which it makes a more intimate contact with the chromite, and consequently produces a better reduction. As the electric furnaces have not a high shaft it is not necessary to use a reducing material of high compressive strength.

TYPES OF FURNACES USED.

There are several types of electric furnace used in the manufacture of ferrochrome. All are constructed on the principle of the Sieman's crucible with a conducting hearth, or are similar to the Héroult steel furnace with vertical electrodes in series. In a conducting-hearth furnace the lining of the walls may be of chromite or, more commonly, magnesite or dolomite tar mixture, with a carbon block in the bottom; or, as in the Girod furnace, it may have a conducting hearth of iron rods buried in the refractory bottom; or, as in the Sieman's furnace, it may have a nonconducting hearth in which the lining of the hearth and walls is entirely of a noncarbon refractory, such as dolomite, magnesite, or chromite. The walls are thin, not over 9 inches thick, but the electrodes are far enough from them to allow the molten charge to freeze on the walls and thus form a highly refractory lining. Some furnaces were water cooled at first to assist this result, but few are now so constructed. Ferrochrome freezes to a certain extent on the bottom, so that molten metal does not have so much chance to come in contact with the carbon bottom.

PROCESS OF MANUFACTURE.

The process of manufacturing ferrochrome consists of mixing a charge of chromite and anthracite coal in proportions based upon the theoretical calculations given. The charge is shoveled into the open top of the furnace and around the electrodes. The furnace is charged at regular intervals so as to keep the top in an unfused condition. The ore is finely powdered, but the coal is about as it comes from the mine. There is some loss of ore that is carried off with the gases that escape at the top of the furnace. At intervals of two to four hours the ferrochrome is tapped off from the bottom into iron pots set on cars, from which it and any slag are dumped when solid. The slag and metal come out of the same tap hole. When cool the metal is broken with hammers to separate it from the slag, and is packed into kegs or boxes. In tapping an iron rod is used to open up the tap hole, which is plugged with fire clay.

Ferrochrome manufacture is carried on continuously, in some cases for two years, before the furnace is shut down for repairs. Ferrochrome containing as low as 5 per cent carbon can be made economically when produced directly from ore in one operation. It has been possible to produce an alloy containing as low as 2 per cent carbon in one operation, but great care in regulation of the charge and operation of the furnace is necessary. The slags from the average ferrochrome furnace making a 5 per cent carbon alloy run from 0.5 to 1 per cent Cr_2O_3 .

The refining of the high-carbon grades of ferrochrome is done in an arc furnace where the alloy is subjected to a prolonged heating with a slag of chromite, lime, and fluorspar, proportioned to the amount of carbon and silicon to be removed and the silica that must be fluxed in the chromite used. The ferrochrome covered by this slag is heated at a high temperature for a length of time dependent on the carbon in the alloy. The resulting slag contains about 25 per cent Cr_2O_3 and is ground up and sent through the smelting furnace again, so that there is not much loss of chromium. By this method it is possible to reduce the carbon from 10 to 0.25 per cent, but the more usual low-carbon grade contains 0.5 per cent carbon. The percentage of carbon in the ferrochrome varies from 9.5 to 0.6 per cent carbon, but the iron increases with the decrease of carbon. This would seem to show that in refining, the iron oxide of the chromite is first reduced, the chromic oxide merely serving to keep a slag with excess chromium, thus preventing any final lowering of the percentage of chromium in the alloy. As the carbon is oxidized so is the silicon until the content is as low as 0.2 per cent, as compared with 2 to 5 per cent in the crude alloy. Aluminum is also considerably eliminated, and phosphorus and sulphur are reduced to a slight degree. Although the market is small for ferrochrome very low in carbon, the difference in price between the 9 per cent alloy and the 0.5 per cent product does not seem warranted. The refining expense is almost entirely the cost of power, and with the cheap power generally available at a ferro-alloy plant the expense is not as high as \$300 per ton, the difference in price of the products mentioned.

Processes based on the use of silicon or some reducing agent other than carbon have not been commercially successful because the cost of such reducing materials as silicon, ferrosilicon, and aluminum is much higher than that of carbon. Also the products may be contaminated with the reducing material just as when carbon is used, so that a subsequent refining is necessary. Carbon-free ferrochrome for high-grade steels has, however, been successfully made by the thermit process, but the amount made in this way is comparatively small. Analyses of various grades of ferrochrome follow.

Analyses of typical ferrochromes.

Component.	Crucible. ^a	Blast furnace. ^a	Electric furnace. ^b				
			With 8 to 10 per cent carbon.	With 7 to 8 per cent carbon.	With 5 to 6 per cent carbon.	With 3 to 4 per cent carbon.	With 1 per cent carbon.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Chromium.....	19.8	41.39	64.5	63.5	64.0	64.0	63.5
Iron.....			22.0	21.5	28.5	31.0	35.0
Carbon.....	3.8	7.12	9.5	7.5	5.5	3.5	.6
Silicon.....		.21	2.25	5.8	.4	.3	.2
Aluminum.....			.80	.8	.5	.4	.10
Manganese.....	.33	1.84	.15	.15	.15	.15	.10
Calcium.....			.25	.25	.25	.30	.35
Sulphur.....			.04	.04	.04	.04	.03
Phosphorus.....		.052	.03	.03	.03	.02	.02

^a Venator, W., Über Eisen Liegierungen und Metall für die Stahl Industrie: Stahl und Eisen, vol. 28, 1908, p. 41.
^b Girod, P., Studies on the electrometallurgy of ferro-alloys and steel: Trans. Faraday Soc., vol. 11, 1911, p. 172.

POWER AND ELECTRODE CONSUMPTION.

The power consumption in a ferrochrome furnace of the Meraker Electric Smelting Co., at Kopperaaen, Norway, was recently given as 3 kilowatt-hours per pound, or 0.68 kilowatt-year per short ton in making a ferrochrome containing 5 per cent carbon. At Kanawha Falls, W. Va., ferrochrome was made in a crucible electric-arc furnace with a power expenditure of 3.6 kilowatt-hours per pound, or 0.72 kilowatt-year per ton.^a This product contained 70.96 per cent chromium, 23.23 per cent iron, 5.21 per cent carbon, 0.5 per cent silicon, 0.008 per cent phosphorus, and 0.078 per cent sulphur. At both Kopperaaen and Kanawha Falls an ore containing about 50 per cent Cr₂O₃ was used. The Kopperaaen ferrochrome contains 65 to 70 per cent chromium. In the experiments of the writer, which have been described, a product containing 50 to 68 per cent chromium and 4.32 to 9.31 per cent carbon was obtained with an ore containing 46.35 per cent Cr₂O₃, 3.02 kilowatt-hours per pound or 0.69 kilowatt-years per ton. A 750-kilowatt furnace of the Alby-carbide type at Kopperaaen, operating continuously, uses on the average about 3 kilowatt-hours per pound of ferrochrome produced, or 0.68 kilowatt-year per short ton, when chromite ore containing 50 per cent Cr₂O₃ is charged; and the product contains 5 per cent or more of carbon and 65 per cent of chromium.

The electrode consumption in ferrochrome manufacture depends somewhat on the proportion of carbon in the ferro-alloy being produced, as, if the carbon of the charge be materially reduced, the electrode will be oxidized by the charge. With an open-top furnace the consumption varies from 50 to 100 pounds per short ton of ferro-

^a Schoel, G. P., Manufacture of ferro-alloys in the electric furnace: Electrochem. Ind., vol. 2, 1904, p. 450.

chrome produced. As most of the ferrochrome furnaces do not use continuously feeding electrodes this item will probably be kept at a much lower figure in the future. At present in many plants one-half to one-third of the electrode is not used, and, unless the works has its own electrode plant in which stumps can be made into new electrodes, this is lost so far as its electrode value is concerned. Aside from the electrode item, the repair and up-keep cost on a ferrochrome furnace is not high. The linings last one to three years without repair.

COST OF MANUFACTURE OF FERROCHROME.

In the table following, estimates are made of the cost of manufacture of a ferrochrome containing 60 per cent chromium and 8 to 10 per cent carbon from chromite ore containing 50 per cent Cr₂O₃. The power figures and other data were gathered by the writer in Europe and the United States during the past year. The calculations are on the basis of two 750-kilowatt furnaces turning out a total of 6 tons of ferrochrome per 24 hours. The Norwegian plant is within 60 miles; the French plant within 200 miles, and the American plant assumed to be within 100 miles of seaboard. Labor conditions are considered to be as found in the various countries.

Cost of production of ferrochrome per ton (2,000 pounds).

Item.	Norway.		France.		United States.	
	Cost of unit.	Total cost.	Cost of unit.	Total cost.	Cost of unit.	Total cost.
3,000 pounds of chromite.....	\$15.00	\$20.50	\$15.00	\$22.50	\$15.00	\$22.50
500 pounds of coal.....	5.00	1.25	6.00	1.50	5.00	1.25
50 pounds of carbon electrodes.....	.04	2.00	.03	1.50	.04	2.00
0.68 kilowatt-year.....	8.30	5.65	18.66	12.70	26.66	17.65
Labor.....	<i>a</i> 1.00	8.33	<i>a</i> .80	6.66	<i>a</i> 1.50	16.60
Repairs.....		4.00		4.00		4.00
Amortization and depreciation at 5 per cent each.....		2.50		2.50		2.50
Interest on \$50,000 at 6 per cent.....		1.50		1.50		1.50
General and packing.....		4.00		4.00		4.00
Total.....		51.73		56.86		72.00

a Minimum for eight hours.

The figures indicate that it costs \$20.27, or 39.2 per cent, more to manufacture ferrochrome in the United States than in Norway. This increase is due to the higher price of power and increased labor cost here. The cheapest figure at which power can be obtained now in the eastern United States is \$26.66 per kilowatt-year for electro-metallurgical purposes. The duty on ferrochrome is to be 15 per cent ad valorem. The 8 to 10 per cent grade sells for \$110 to \$130 per short ton (see table following).

SELLING PRICES OF FERROCHROME.

Ferrochrome is sold on the basis of a chromium content of 60 per cent, the price varying with the percentage of carbon. As the percentage of chromium varies from a given grade of alloy, the price, based on the carbon percentage, is decreased or increased a certain amount for each per cent variation from 60 per cent chromium. This feature is shown in the following tabulation of prices current in Germany and the United States on January 1, 1913:

Prices of ferro-alloys and metals.^a
GERMANY, F. O. B. DUISBURG.

Alloy.	1912			1913
	Oct. 1.	Nov. 1.	Dec. 1.	Jan. 1.
Ferrosilicon:				
Made in the blast furnace, basis 10 per cent silicon, 2,240 pounds, scale ±\$0.96.....	\$32.90	\$34.90	\$35.40	\$36.80
Made in the electric furnace, basis 45 per cent silicon, 2,240 pounds, scale ±\$1.51.....	73.00	70.00	68.00	70.00
Made in the electric furnace, basis 75 per cent silicon, 2,240 pounds, scale ±\$1.65.....	129.00	126.00	126.00	126.00
Ferromanganese silicon, electric furnace, 2,240 pounds:				
50 to 55 per cent manganese, 23 to 28 per cent silicon.....	112.00	112.00	112.00	112.00
63 to 75 per cent manganese, 20 to 25 per cent silicon.....	112.00	106.50	106.50	106.50
50 to 55 per cent manganese, 30 to 35 per cent silicon.....	117.50	117.50	117.50	117.50
Ferromanganese, basis 80 per cent manganese, 2,240 pounds, scale ±\$0.55.....	61.90	61.90	64.25	68.00
Ferrochrome:				
Carbon free, thermit process, basis 60 per cent chromium, 2,000 pounds, scale \$8.95.....	500.00	500.00	550.00	550.00
Electric furnace—				
Refined No. I, 0.3 to 0.75 per cent carbon, basis 60 per cent chromium, scale \$8.95, 2,000 pounds.....	436.00	436.00	436.00	436.00
Refined No. II, 1 to 2 per cent carbon, basis 60 per cent chromium, scale \$7, 2,000 pounds.....	262.00	262.00	262.00	262.00
4 to 6 per cent carbon, basis 60 per cent chromium, scale \$4.20, 2,000 pounds.....	105.00	112.25	118.50	133.00
Ferrotitanium (10 to 15 per cent titanium), per pound.....	.188	.188	.188	.188
Ferrotungsten, 85 per cent tungsten, 0.5 to 1 per cent carbon, per pound of tungsten contained.....	.75	.71	.75	.765
Ferromolybdenum, 70 to 80 per cent molybdenum, per pound of molybdenum contained.....	2.06	2.06	2.06	2.06
Nickel, 98 to 99 per cent nickel, per pound.....	.43	.43	.43	.43
Aluminum, 98 to 99 per cent aluminum, per pound.....	.20	.206	.213	.218
Tungsten metal, 96 to 98 per cent tungsten, per pound.....	.70	.72	.725	.72

UNITED STATES, F. O. B. PITTSBURGH, PA.

Ferromanganese, basis 80 per cent manganese, 2,240 pounds.....				\$63.30
Ferrosilicon, blast furnace, f. o. b. Ashland, Ky.:				
10 per cent silicon.....				24.00
11 per cent silicon.....				25.00
12 per cent silicon.....				26.00
Ferrosilicon, electric furnace, 50 per cent silicon, 2,240 pounds.....				75.00
Ferrotitanium, 10 to 15 per cent titanium, f. o. b. Niagara Falls, N. Y., per pound in carloads.....				.08
Ferrovandium, per pound of vanadium content.....				2.50

^a Stahl und Eisen, vol. 33, 1913, p. 85 (revised).

USES OF FERROCHROME.

Ferrochrome is used extensively in the manufacture of steel for armor plates, armor-piercing projectiles, wire, bullet-proof steel, tool steel, high-speed steel, high-grade castings, stamp-mill shoes and

dies, safe steel, tires, axles, springs, razors, file and cutlery steel, and for many other purposes. As shown in the table on page 138, electric furnace ferrochrome is made in five grades, of which those containing 4 per cent carbon are classed as refined, the carbon in the others being regulated in the charge to a certain extent.

Ferrochromes containing less than 2 per cent carbon are used in the manufacture of tool steels and high-speed steels. The alloy containing 3 to 4 per cent chromium is used principally in the manufacture of steel castings. For general open-hearth casting ferrochrome containing 4 to 6 per cent of carbon is used.

For the manufacture of armor plate and projectiles, the high-carbon ferrochromes containing 8 to 9 per cent carbon are generally used. This alloy is much cheaper and also its use is advisable for armor plate, because the carbon must be added at some time to harden the steel. A ferrochrome containing a high proportion of carbon used in the open-hearth furnace will give a yield much superior to that containing a low proportion of carbon, because if the metal is still slightly oxidized, the oxide will act on the carbon rather than on the chromium. For the purpose of protecting the chromium from oxidation, armor-plate manufacturers use a ferrochrome that contains a high percentage (4 to 5 per cent) of silicon. In this way the oxides act on the silicon before they act on the chromium.

The distinguishing feature of chrome steels is their great hardness.^a Chrome steels usually contain 1 to 2 per cent chromium and 0.8 to 2 per cent carbon. Armor plate contains about 3.25 per cent nickel, 1.5 per cent chromium, and 0.25 per cent carbon. In armor plates the nickel and chromium are sometimes added in the form of a ferrochrome-nickel containing 52 per cent chromium, 18 per cent nickel, 28.96 per cent iron, 0.5 per cent carbon, 0.5 per cent silicon, 0.03 per cent sulphur, and 0.01 per cent phosphorus.

FERROMANGANESE.

HISTORY.

In steel manufacture more ferromanganese than any other ferroalloy is used. The industrial importance of ferromanganese began in 1866 with the introduction of the Bessemer process, which required a strong deoxidizing agent to produce a sound steel. In 1866, Prieger of Bonn, Germany, made in the crucible some ferromanganese containing 70 to 80 per cent manganese. At Terre-Noire, France, a little later it was made in the open-hearth furnace and crucible in a combination process. The first ferromanganese to be made in the blast furnace was produced in Sweden in 1873, and contained 33 per cent manganese. In 1875, at Terre-Noire, Pourcel made in the blast furnace some ferromanganese that contained 75 to 80 per cent

^a Stoughton, Bradley, The metallurgy of iron and steel, 1908, p. 399.

manganese. Since that time up to a few years ago all ferromanganese was made in the blast furnace. At present a comparatively small quantity is made in the electric furnace, the greater part of that so made being used for the production of ferromanganese-silicon or silicomanganese. When the iron-manganese alloy contains less than 25 per cent manganese it is called spiegeleisen; when it contains more than 25 per cent it is called ferromanganese.

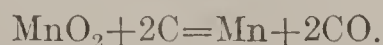
EXPERIMENTS IN THE PRODUCTION OF FERROMANGANESE.

Stassano ^a in 1908 made some experiments in the production of ferromanganese in a 75-kilowatt electric furnace of the type of the Stassano steel furnace. The charge consisted of 1,000 parts of manganese ore, 300 parts of charcoal, 60 parts of lime, and 80 parts of 25 per cent sodium silicate used as a binder for briquets of the above composition, the form in which the charge was smelted. The ore used contained 45.65 per cent Mn_3O_4 , 16.1 per cent Fe_2O_3 , 3.05 per cent Al_2O_3 , 30.16 per cent SiO_2 , 0.15 per cent BaO , 1.2 per cent CaO , 0.43 per cent MgO , 0.817 per cent S, and 0.34 per cent P. The desired product was to be silicomanganese containing about 60 per cent manganese and 20 per cent silicon. The charge was submitted to the heat radiated from the arc as in all Stassano furnaces.

The product contained 17.76 per cent iron, 17.6 per cent silicon, 62 per cent manganese, 1.8 per cent carbon, 0.028 per cent phosphorus, and a trace of sulphur. No data showing the losses of manganese were given. The energy consumption was 3.86 kilowatt-hours per pound, or 7,560 kilowatt-hours, 0.86 kilowatt-year, per long ton.

THEORY OF PRODUCTION.

Ferromanganese is manufactured from manganese oxide ore with carbon as a reducing agent. Because of the high reduction point of manganese dioxide, $1,105^\circ \text{C}$., considerably more coke is necessary in the blast-furnace charge than in the iron blast furnace. Reduction takes place in the electric furnace according to the reaction—



For the reduction of 100 parts of manganese, 43.5 parts of carbon are necessary.

Carnot ^b and Goutal have shown the existence of the following double carbides of iron and manganese in ferromanganese of different percentages of carbon:

Double carbide.	Percentage of manganese.
$\text{Fe}_3\text{C} \cdot 4 \text{Mn}_3\text{C}$	74 to 85
$\text{Fe}_3\text{C} \cdot 2 \text{Mn}_3\text{C}$	60 to 74
$2\text{Fe}_3\text{C} \cdot \text{Mn}_3\text{C}$	30 to 60
$4\text{Fe}_3\text{C} \cdot \text{Mn}_3\text{C}$	below 18

^a Stassano, E., Treatment of iron and steel in the electric furnace: *Electrochem. and Met. Ind.*, vol. 6, 1908, p. 315.

^b Guillet, L., *Étude industriel des alliages métalliques*, p. 405.

The single carbide of manganese is Mn_3C , similar to Fe_3C , the carbide of iron.

Ferromanganese usually contains about 6 per cent carbon, when it is silvery white. When the silicon content in the alloy is low, the carbon is all in the combined state. With more than 5 per cent silicon the carbon falls to about 2 per cent, and as the silicon increases the carbon in the alloy decreases.

PROCESS OF MANUFACTURE.

Ferromanganese is still manufactured almost entirely in the blast furnace. When the manufacture of ferro-alloys in the electric furnace began, attempts were made to make ferromanganese. These failed, chiefly because of operation of the electric furnace at so high a temperature as to produce great volatilization of the manganese. As the electric furnace became better understood, heating by resistance only was found possible, thus insuring a low enough temperature.

Most of the blast-furnace ferromanganese used in this country, with the exception of that made by one large corporation, is made in England. Recently some German ferromanganese has been put on the market. Russian or Turkish manganese oxide ore is used. Ores high in silicon can not be used because of the high loss of manganese as silicate in the slag. The results of a typical blast-furnace run, including figures as to the ore used, and the slag and the alloy produced, are given in the table below. In this run the average amount of coke consumed was 2.37 long tons per long ton of ferromanganese produced. Of the total manganese charged 76.9 per cent was saved and 23.1 per cent lost. Of the total slag lost 6.7 per cent was in the slag and 18.4 per cent was volatilized or lost as dust.

Results of blast-furnace run to produce ferromanganese.^a

Ore.		Slag.		Ferromanganese.	
Constituent.	Percentage.	Constituent.	Percentage.	Constituent.	Percentage.
Mn.....	51.6	SiO_2	30.32	Mn.....	80.20
Fe.....	1.45	FeO.....	1.41	Fe.....	11.80
P.....	.173	MnO.....	8.52	Si.....	1.16
SiO_2	7.80	Al_2O_3	10.88	P.....	.38
Al_2O_3	1.12	CaO.....	41.34	C.....	6.46
CaO.....	1.80	MgO.....	2.96		
MgO.....	2.18	P_2O_501		
BaO.....	2.20	CaS.....	3.94		
		BaO.....	.48		

^a Jakoki, J., Ferromangen in Hochofen; Stahl und Eisen, vol. 29, 1909, p. 1191.

The loss of manganese by volatilization, or as dust, varies between 15 and 30 per cent of the total quantity of manganese charged. The average total loss is about 30 per cent, of which 10 per cent goes to the slag and 20 per cent is volatilized or lost as dust. Any phosphorus

in the ore goes to the metal, but there is no difficulty in slagging the sulphur. No more than 0.02 per cent of phosphorus should be present for each 10 per cent of manganese in the ore. The manganese as a rule is unevenly distributed through the alloy. Usually some limestone is necessary in the charge. The alloy produced in the run discussed above is high in phosphorus. An average English ferromanganese contains 83.4 per cent manganese, 9.04 per cent iron, 6.5 per cent carbon, 0.8 per cent silicon, 0.25 per cent phosphorus, and 0.01 per cent sulphur.

When ferromanganese is being made in the electric furnace, the electrodes and voltage are regulated so as to prevent arcking. With a mixture of 813 parts of ore containing 30 per cent manganese, 178 parts of anthracite coal (4 to 5 per cent ash), and 90 parts of fluorspar, about 450 pounds of product was obtained per ton of mixture treated. The product contained 85 per cent ferromanganese.^a Of the total manganese charged 26.8 per cent was lost by volatilization, as dust, and in the slag. Ferromanganese made in the electric furnace has considerably lower carbon content than the blast-furnace ferromanganese, and is used in the molten state by the ferro-alloy manufacturer for making silicomanganese.

USES OF FERROMANGANESE.

The greater part of the ferromanganese manufactured is used in the deoxidation and recarburization of ordinary Bessemer and open-hearth steel. It is a strong deoxidizing agent and also assists in the elimination of sulphur from steel. In foundry work ferromanganese is used for softening, strengthening, and purifying hard or chilling iron. It is also employed for strengthening soft iron.

The next greatest use for ferromanganese is as a fixed addition in the production of manganese steel. When the manganese content of steel is more than 1 per cent, the metal becomes hard and somewhat brittle, and these qualities increase in intensity with every increase of manganese up to 4 to 5.5 per cent when the steel can be powdered under the hammer.^b With 7 per cent manganese new properties appear, which are well marked in steel having a manganese content of 10 per cent, and reach a maximum with a manganese content of 12 to 15 per cent. Manganese steel usually contains 12 to 13 per cent of manganese and 1.5 to 2 per cent of carbon. With that proportion of manganese the strength and durability of the metal is at its maximum. When cooled slowly after casting manganese steel is almost as brittle as glass. It must be reheated and quenched at a high temperature in water. After such treatment it is as ductile as soft carbon steel or wrought iron, and its tensile strength is about three times

^a Guillet, L., *Étude industriel des alliages métalliques*, p. 405.

^b Stoughton, B., *The metallurgy of iron and steel*, 1911, p. 397.

as great. The hardness of the product after the latter treatment is almost the same as before, being so hard that machining is not practical. The forging and working of manganese steel is very difficult and is understood by only a few.

Manganese steel is used for the jaws and wearing parts of rock-crushing machinery, for railroad frogs and crossings, for railroad rails on curves, for mine-car wheels, and for burglar-proof safes. Because of its great hardness without brittleness its life for the uses mentioned is many times that of all other steels.

FERROMANGANESE-SILICON.

Ferromanganese-silicon or silicomanganese is an alloy developed since use has been made of the electric furnace for ferro-alloy manufacture. Ferromanganese is made from manganese ore in an electric furnace operated as a resistance furnace, and ferrosilicon is made in a separate arc furnace. The two products are mixed while hot in a ladle, to form the alloy of the desired percentage of manganese and silicon. Ferromanganese containing about 85 per cent of manganese is made. The percentage of ferrosilicon depends on the amount of iron desired in the finished silicomanganese. The other method of manufacture is to mix coal, quartz, manganese, oxide ore, and iron turnings in the proper proportion to form the desired silicomanganese and smelt it all in one furnace which must be kept at a high temperature because of the high reduction point of silica. The great disadvantage of this method is the high loss of manganese by volatilization. This method, however, seems to be the favorite one, as it is quicker, only one furnace is involved in the process, and less labor is necessary. It should be noted from the table on page 140 that the price of silicomanganese does not increase with the manganese content but with an increase in the percentage of silicon. This is because much more energy is necessary for reduction of silica than of manganese oxide. A third method of manufacture is to reduce rhodonite (MnSiO_3) with carbon in the electric furnace. In this method coal or coke is mixed with rhodonite containing 38 per cent manganese, in the proportion of 600 parts of ore to 100 parts of coke. The electrode consumption is very high, being about 200 pounds per ton of product, and the power consumption is 5,400 kilowatt-hours or 0.62 kilowatt-year per ton. Keller adds ferrosilicon to a fused mixture of ferrosilicon, quartz, carbon, and manganese ore, claiming that the reduction temperature remains low and prevents volatilization of manganese.

Three grades of ferromanganese-silicon are made as follows: 50 to 55 per cent manganese, 23 to 28 per cent silicon; 68 to 75 per cent manganese, 20 to 25 per cent silicon; 50 to 55 per cent manganese, 30 to 35 per cent silicon. The carbon content in these alloys is low,

owing to the action of silicon in breaking up the combined carbon in the ferromanganese, causing the carbon to separate as graphite, which rises to the surface of the bath. With less than 16 to 18 per cent silicon in the alloy, carbon remains in the combined state; with more than that percentage, any carbon present is in the form of graphite. Analyses of typical silicomanganese are given below:

Analyses of typical ferromanganese-silicon.

Constituent.	Analysis 1.	Analysis 2.	Analysis 3.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Silicon.....	24.1	24.6	24.1
Manganese.....	74.2	70.3	55.0
Iron.....	.77	3.8	19.0
Aluminum.....		.4	.4
Calcium.....		.3	.3
Magnesium.....		.2	.2
Carbon.....	.30	.35	.35
Sulphur.....	.01	.02	.02
Phosphorus.....	.02	.04	.04

Alloys similar to those represented in analyses 1 and 2 are used for incorporating manganese and silicon in steel. The alloy is also used for deoxidizing, when a composition similar to that given in analysis 3 is generally employed. The product has the advantage over ferromanganese of being low in carbon, yet possessing as strong a deoxidizing power. Its use is especially advantageous in making high-grade manganesesilicon steel. Silicomanganese is usually added to the furnace just before the furnace is tapped, or in the casting ladle during tapping.

ELECTRIC SMELTING OF MOLYBDENITE AND THE PRODUCTION OF FERROMOLYBDENUM.

HISTORY.

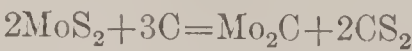
Ferromolybdenum is generally manufactured in the electric furnace from the raw sulphide ore of molybdenum, molybdenite. It is also made by the reduction of the roasted sulphide with carbon in a crucible or an electric furnace. The alloy is not widely used because of its high cost, which is caused by the irregularity of the ore supply. With a better and steadier source of ore, ferromolybdenum would probably be used in place of ferrotungsten. Ferromolybdenum was made from roasted ore in the crucible before the introduction of the electric furnace, but has been produced commercially directly from the sulphide only since 1900, following the use of the electric furnace.

INVESTIGATORS.

Investigators of the electric smelting of molybdenite or of the production of ferromolybdenum are mentioned below, their methods being briefly outlined.

GUICHARD.

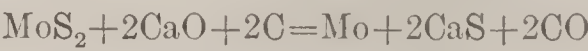
Guichard,^a working with Moissan between 1890 and 1895, obtained a crude molybdenum from natural molybdenite by reduction with carbon in the electric furnace. This product was really a carbide, as it contained 91.8 per cent molybdenum, 2.1 per cent iron, and 6.64 per cent carbon. The operation was performed according to the following reaction:



The difficulty in the process is to obtain pure metal from impure ores, because there is a tendency for any impurities to concentrate in the metal.

LEHNER.

Lehner^b conducted a series of experiments on the production of a ferromolybdenum free from sulphur, using molybdenite in the electric furnace. After several unsuccessful experiments, in which the oxides of other metals were used to decompose molybdenite or in which the reducing agent was aluminum, he tried to accomplish the reduction in the electric furnace with a very basic slag, according to the following reaction:



The results of some of his experiments are tabulated below:

Results of experiments of Lehner.

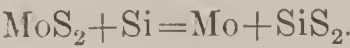
Experiment No.....	1	2	3	4	5
Charge:					
MoS ₂pounds..	3.3	3.3	1.23	5.5	5.5
Fe ₂ O ₃do.....			2.11	4.73	4.73
CaO.....do.....	2.31	3.08	.86	^a 9.16	^a 9.16
Coke.....do.....	.55	.85	.79	2.18	2.18
Product:					
Molybdenum.....per cent..	90.7	88.3	43.30	50.33	48.72
Iron.....do.....	2.9	2.89	49.8	45.95	46.56
Carbon.....do.....	4.1	5.60	6.27	1.70	1.06
Silicon.....do.....		2.95	.31	1.51	2.61
Sulphur.....do.....	.86	.04	.05	.10	.13

^a CaCO₃.

These results show the possibilities of sulphur elimination by means of lime in the electric furnace, but do not show where the molybdenum losses occur—whether in the slag or by volatilization. An extraction of 85 per cent was made in experiment 2.

NEUMANN.

Neumann^c attempted reduction of molybdenite according to the reaction:



^a Guichard, M., Sur la molybdénite et la préparation au molybdene: Compt. Rend., vol. 122, 1896, p. 1270.

^b Lehner, W., Herstellung von Molybdenum: Metallurgie, vol. 3, 1906, p. 549.

^c Neumann, G., Herstellung von Ferroliegierung im electrischen Ofen: Stahl und Eisen, vol. 28, 1908, p. 356.

He obtained unsatisfactory results. The metal produced contained 2.87 per cent iron, 2.06 per cent silicon, and 13.89 per cent sulphur, the remainder being molybdenum. Upon this method, several patents issued to Becket are based.

EXPERIMENTS ON REDUCTION OF MOLYBDENITE.

An experiment was made by the writer,^a who used the electric furnace described in the discussion of the ferrochrome experiments to test the possibility of sulphur elimination with a charge of molybdenum and excess lime, carbon being used as a reducing agent. The experiment was made directly after some experiments on the production of molybdenum steel directly from hematite and molybdenite, and can not be considered as quantitative, as the furnace lining was in poor condition. Some iron left in the furnace diluted the product. The electric furnace was regulated so that all heating was through resistance only. On tapping, 1.5 pounds of ferromolybdenum was obtained. The odor of sulphur was noticeable during the experiment. The product was brittle, but was not extremely high in sulphur in view of the amount of sulphur in the charge. There was 30 per cent sulphur in the ore, but only 0.19 per cent sulphur in the product. A commercial ferromolybdenum having 50 per cent of molybdenum contains about 0.03 per cent sulphur and 0.5 to 4 per cent carbon. The results of the experiment are shown in the table below. The coke, fluorspar, and lime had the composition given in the table on page 128.

Production of ferromolybdenum from molybdenite.

Ore.		Charge.		Metal.		Slag.	
Constituent.	Percentage.	Constituent.	Pounds.	Constituent.	Percentage.	Constituent.	Percentage.
Mo.....	47.5	Molybdenite	4.0	Mo.....	50.55	MoO ₂	3.18
Fe.....	1.23	Coke.....	1.04	C.....	.73	CaO.....	41.80
SiO ₂	13.90	Lime.....	3.01	Si.....	.94	MgO.....	12.75
P.....	.608	Fluorspar...	.25	P.....	.078	FeO.....	7.56
S.....	30.64			S.....	.19		

From this experiment no conclusion can be drawn as to the loss of molybdenum, but in three runs made on the production of molybdenum steel from hematite and molybdenite with carbon as a reducing agent, the average total molybdenum loss was 46.1 per cent, of which 25.4 per cent was in volatilization and 20.7 per cent by slagging. Thus 56.2 per cent of the total molybdenum loss was by volatilization.

The conclusions made from the ferromolybdenum experiment were: First, that ferromolybdenum low in carbon can be made directly from molybdenite in the electric furnace, with excess lime as a desulphur-

^a Keeney, R. M., The production of steel and ferro-alloys directly from ore in the electric furnace: Iron and Steel Inst., Carnegie Scholarship Memoirs, vol. 4, 1912, p. 108.

izing agent and carbon as a reducing agent; second, that a product with a low percentage of carbon can be made; and, third, that sulphur can be readily slagged as calcium sulphide with a charge of excess lime.

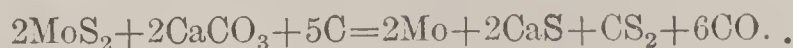
THEORY.

A ferromolybdenum containing 80 per cent molybdenum has a dull iron gray color, coarse structure, a high density, and is nonmagnetic. It does not break up easily. The 75 per cent alloy is about the same as the above, but has a needle structure, and when treated with another magnet is so magnetic that it has about three times the magnetism of soft steel.

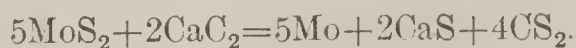
A commonly used basis for calculating the charge for the reduction of ferromolybdenum from molybdenite is the reaction used by Becket in his patent.^a Carbon is the reducing agent and lime is used as a desulphurizing agent. The reaction is as follows:



Pure molybdenite contains 60 per cent molybdenum and 40 per cent sulphur. According to the above reaction, 100 parts of molybdenum are reduced from 170 parts of molybdenite by 18.8 parts of carbon. If the reaction is to be complete, for every 100 parts of molybdenum present as the molybdenite, 58 parts of lime are necessary for the slagging of the sulphur. If calcium carbonate instead of lime be used, the reaction is:

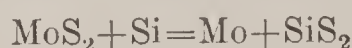


If calcium carbide be used as a reducing agent, the following reaction takes place:



Becket, in his patent, states that an alloy containing less than 0.2 per cent carbon can be made by this method. Molybdenum does not combine as readily with carbon to form carbides as do some of the other metals, such as chromium and tungsten.

Becket was granted a patent^b based upon the desulphurizing of molybdenite with silicon. The reaction involved is:



Neumann's experiments indicate that the use of the latter reaction is not very satisfactory. By this method the production of 100 parts of molybdenum requires 29 parts of silicon, which would probably be used in the form of ferrosilicon, the iron in the ferrosilicon making the desired percentage in the ferromolybdenum.

^a U. S. Patent 835052, Nov. 6, 1906.

^b U. S. Patent 855157, May 28, 1907.

PROCESS OF MANUFACTURE.

Ferromolybdenum is manufactured commercially in an electric furnace of the electrode type operated as a resistance furnace, or in a crucible furnace of the resistance type. Comparatively little ferromolybdenum is made because of difficulty in getting ores, as satisfactory methods of concentrating molybdenite ores have not as yet been fully developed. There are deposits of molybdenite in the United States in Maine, Oregon, Colorado, Nevada, and Washington. There are also deposits in Canada, Germany, Japan, Mexico, New South Wales, New Zealand, Norway, Peru, and Queensland. Queensland and New South Wales are the largest producers. Buyers require an ore or a concentrate containing 90 to 95 per cent molybdenite, for which the price is about \$450 per short ton. In commercial manufacture, raw or roasted molybdenite, iron turnings, lime, and coke or coal are mixed to give a ferromolybdenum of the desired proportions, and the mixture is treated in the electric furnace. The furnaces are operated intermittently; that is, one charge is completely discharged before another is added. Analyses of typical electric-furnace ferromolybdenum are given in the table following. Ferromolybdenum is first made as a high-carbon alloy containing 3 to 4 per cent carbon. This is then decarburized with a lime slag or a slag of lime and iron oxide. If the latter is used, the percentage of iron in the alloy is increased.

Analyses of typical ferromolybdenum.

Analysis No.....	1	2	3	4
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Molybdenum.....	85.8	75.0	85.2	50.311
Iron.....	10.963	18.5	14.047	48.92
Carbon.....	3.07	4.0	.45	.35
Silicon.....	.11	.2	.252	.30
Aluminum.....		.1		
Calcium.....		.15		
Manganese.....		.15		
Sulphur.....	.05	.03	.031	.030
Phosphorus.....	.007	.03	.02	.20

USES.

Ferromolybdenum in definite proportions is added to steel in the open-hearth crucible or in the electric furnace. It gives the steel properties similar to those of tungsten steel, but only one-third to one-quarter as much is necessary. Molybdenum is added to steel that is to be used for such forgings as crank and propeller shafts, for gun barrels, wire, boiler plates, armor-piercing shells, motor-car steel, magnet steel, and high-speed steel. Molybdenum considerably increases the elongation and elastic limit of steel. By the addition of 0.25 per cent of molybdenum the

elongation has been increased from 4 per cent up to 45 per cent. The addition of small quantities of molybdenum to nickel steels increases their resistance to blows without proportionally diminishing their elongation. A nickelchrome molybdenum steel has been used for armor plate.

Tempering has a much more energetic action in molybdenum steel than in carbon steels. If sufficient chrome is added to steels containing a high percentage of molybdenum, they lose a large part of their electricity, harden in the air, and have the same qualities as tungsten high-speed steels. The advantage in using molybdenum instead of tungsten in a high-speed steel is that it gives all the desirable qualities to high-speed steel, with the advantage that the carbon content is kept below 1 per cent, the hardness and brittleness of tungsten high-speed steels being thus avoided. Also, if ferromolybdenum could be produced as cheaply as ferrotungsten it would be more economical to use because of the decreased amount necessary to give the same qualities.

FERRONICKEL.

Nickel is usually incorporated in steel by the addition of metallic nickel rather than of ferronickel. Most of the ferronickel now used occasionally is made in a crucible furnace, fired by coal or gas, rather than in the electric furnace, as it is made by simply melting iron and nickel in the proper proportions. The metallic nickel used contains 99 per cent nickel. Ferronickel as supplied contains 25 per cent, 35 per cent, 50 per cent, 75 per cent, and 85 per cent nickel. The impurities in these alloys consist of 0.5 to 1 per cent carbon, 0.2 to 0.3 per cent silicon, 0.01 to 0.02 per cent sulphur, and 0.2 to 0.3 per cent phosphorus. It is claimed that these alloys give a more homogeneous mixture of nickel in the steel than is obtained by the use of the ordinary nickel metal. The alloy is malleable, homogeneous, and can be readily rolled, drawn, or worked. The ferronickel containing 25 per cent nickel is practically nonmagnetic.

EXPERIMENTS ON THE PRODUCTION OF NATURAL ALLOYS OF NICKEL DIRECTLY FROM ORE IN THE ELECTRIC FURNACE.

EXPERIMENTS AT SAULT STE. MARIE.

In 1907, during the course of the experiments on the production of pig iron in the electric furnace at Sault Ste. Marie, several runs were made on the reduction of roasted pyrrhotite with charcoal.^a

^a Haanel, E., Report on the experiments made at Sault Ste. Marie, Ontario, under Government auspices, in the smelting of Canadian iron ores by the electrothermic process: Mines Branch, Department of the Interior, Canada, 1907.

The furnace used in these experiments was similar in general design to that shown in figure 38. It took about 150 to 175 kilowatts. The results of the experiments are given in the table below:

Results of experiments on the production of ferronickel from roasted pyrrhotite.

Roasted pyrrhotite.		Charcoal.		Limestone.		Charge.	
Constituent.	Per cent.	Constituent.	Per cent.	Constituent.	Per cent.	Constituent.	Pounds.
SiO ₂	10.96	Moisture.....	2.20	SiO ₂	1.71	Ore.....	14,500
Fe ₂ O ₃	65.43	Volatile matter...	20.60	Fe ₂ O ₃ +Al ₂ O ₃81	Charcoal..	4,050
{ (45.8 Fe)		Fixed carbon.....	74.40	CaCO ₃	92.85	Limestone	3,185
Al ₂ O ₃	3.31	Ash.....	2.80	MgCO ₃	4.40	Quartz....	40
CaO.....	3.92			P.....	.004	Fluorspar.	25
MgO.....	3.53			S.....	.052		
S.....	1.56						
P.....	.016						
Cr.....	.41						
Ni.....	2.23						
Average analysis of ferronickel.				Average analysis of slag.			
Constituent.		Per cent.		Constituent.		Per cent.	
Nickel.....		4.01		SiO ₂		21.43	
Copper.....		.71		Al ₂ O ₃		11.60	
Carbon.....		3.19		CaO.....		49.63	
Manganese.....		.10		MgO.....		8.78	
Silicon.....		4.90		MnO.....		.10	
Phosphorus.....		.043		FeO.....		.28	
Sulphur.....		.005		CaO.....		.015	
				NiO.....		.018	
				P ₂ O ₅		4.46	
				S.....		.02	
Length of run, hours and minutes.....							56 20
Mean volts on furnace.....							36.12
Mean amperes on furnace.....							5,000
Power factor.....							.919
Kilowatts.....							165.9
Ferronickel tapped, pounds.....							7,336
Kilowatt-hours per pound.....							1.27
Kilowatt-years per ton.....							.29

The furnace was later used in the production of ferronickel. The results of the work done later is given in the report by the metallurgist of the company that purchased the furnace. The ore and raw materials were of about the composition given for those substances in the table above. Four hundred pounds of briquetted roasted ore, 140 to 150 pounds of limestone, and 120 pounds of charcoal were the proportions used. During four months of operation, 154 short tons of ferronickel of approximately the composition given was produced, with a power consumption of 0.34 kilowatt-year per ton. The electrode consumption was 40 pounds of amorphous carbon per ton of ferronickel. To produce a product containing 4 per cent nickel, 2.75 per cent silicon, 0.8 per cent copper, 0.01 per cent sulphur, and 0.03 per cent phosphorus, on the average 2 tons of roasted pyrrhotite (2 per cent sulphur), 1,500 pounds of limestone, and 1,200 pounds of charcoal were used.

The results of the operation of the furnace show the possibility of producing a low-grade ferronickel directly from roasted ore in the electric furnace. Owing to the low percentage of nickel in the ore a high percentage of nickel could not be concentrated in the metal. There was difficulty in selling the product, so that after a short period of operation the work was discontinued. The product was not pure enough or high enough grade for steel manufacture.

EXPERIMENTS AT PLANT IN NORTH CAROLINA.

A plant was erected at Webster, Jackson County, N. C., where experiments were conducted on the reduction of garnierite in the electric furnace with coke as a reducing agent.^a The furnace used was similar to the one shown in figure 41. The ores used were garnierite, containing 7.3 per cent Ni., 37.5 per cent SiO₂, 29 per cent Al, 1.8 per cent Fe, 10 per cent Mg, 0.1 per cent CO, and 11.9 per cent H₂O, and dunite, containing 1.7 per cent Ni, 41.8 per cent SiO₂, 6.7 per cent Al, 8.3 per cent Fe, 28.2 per cent Mg, 0.02 per cent CO, and 10.9 per cent H₂O. About 10 per cent of coke was used. The product made was a nickel silicide containing 10 to 30 per cent nickel, 20 to 30 per cent silicon, 40 to 50 per cent iron, 5 to 10 per cent Al, 3 to 5 per cent Cr, and 3 to 4 per cent of carbon, magnesium sulphur, and phosphorus. The average power on the furnace was 200 kilowatts. About 7,800 pounds of ore was smelted per 24 hours.^b In a typical run 6.3 kilowatt-hours was used per pound of nickel silicide produced. The silicide contained 11.9 per cent nickel. The plant is no longer in operation. Power was very expensive at the point of erection of the experimental plant. It was stated to cost \$200 per kilowatt-year.

EXPERIMENTS OF STEPHAN.

In 1903, at the works of the Société Anonyme Électrométallurgique Procédés Paul Girod, Stephan worked on the production of a nickel silicide from a silicon ore of nickel.^c The product contained 30 per cent nickel and 47.20 per cent silicon. In 1911, experiments were performed at the same works in an attempt to make a ferronickel containing 3 to 5 per cent silicon.

The furnace used was of the type shown in figure 43. It was of 220 kilowatts capacity and treated 8 tons of ore, with the production of 770 pounds of metal, in 24 hours. The power consumption was 1,760 kilowatt-hours per ton of ore treated, or 6.86 kilowatt-hours per pound of alloy obtained. In practice it is believed the power con-

^a Anon., Electric smelting of nickel ore: *Met. and Chem. Eng.*, vol. 8, 1910, p. 277.

^b Morrison, W. L., Electric-furnace treatment of nickel ore and the development of a commercial process: *Trans. Am. Electrochem. Soc.*, vol. 20, 1911, p. 315; *Met. and Chem. Eng.*, vol. 4, 1911, p. 546.

^c Stephan, M., Einiges über die Erzeugung von Metallen im elektrischen Ofen: *Metal und Erz*, vol. 1, 1912, p. 11.

sumption could be reduced to 1,200 kilowatt-hours per ton of ore, or, on the basis of the same percentage extraction, to 4.69 kilowatt-hours per pound of alloy. The results of the experiment are shown below:

Results of Stephan's experiments on the production of ferronickel.

Ore.		Ferronickel.		Slag.	
Constituent.	Per cent.	Constituent.	Per cent.	Constitu-ent.	Per cent.
NiO.....	8.33	Nickel.....	41.50	NiO ₂	0.31
SiO ₂	28.58	Iron.....	51.6	SiO ₂	53.98
Fe ₂ O ₃	14.61	Silicon.....	4.33	FeO.....	1.56
MgO.....	20.65	Silicide.....	Trace.		
Al ₂ O ₃	2.33	Aluminum.....	.81		
H ₂ O.....	12.32	Carbon.....	1.34		
Moisture.....	12.54	Sulphur.....	.04		

FERROPHOSPHORUS.

Ferrophosphorus is made in the electric furnace from apatite or other phosphorus-bearing mineral. It is used in the open hearth for enriching the slag so as to get phosphorus into the steel, as steel for sheet purposes, particularly tin plates, can be rolled more satisfactorily if it contains phosphorus. Two grades of ferrophosphorus are used. One contains 16 to 20 per cent phosphorus and has the following analysis: 17.5 per cent phosphorus, 76.2 per cent iron, 0.42 per cent silicon, 0.27 per cent carbon, and 5.75 per cent manganese. The other grade contains 20 to 25 per cent phosphorus and has 24.0 per cent phosphorus, 73.3 per cent iron, 2.47 per cent silicon, 0.03 per cent carbon, 0.08 per cent sulphur, and 0.10 per cent manganese. Alloys containing phosphorus and manganese are also made. Some ferrophosphorus is made in Canada in the electric furnace, but most of it comes from Europe, except what is produced by a blast furnace in Tennessee.

FERROSILICON.

HISTORY.

Ferrosilicon is the most extensively used of all the ferro-alloys produced in the electric furnace. If all ferro-alloys be considered, it is second only to ferromanganese in amount consumed. Ferrosilicon was first prepared in 1810 by Berzelius ^a, who, with a mixture of iron shavings, silica, and carbon, obtained an alloy containing 9 per cent silicon. In 1811, Strohmayer ^a repeated the experiments of Berzelius and prepared a series of alloys containing 2.2 to 9.3 per cent silicon. Valten ^b made ferrosilicon at Terre-Noire in 1872 by the reduction of iron oxide and silica in a crucible with carbon. There was 10 to 12 per cent sili-

^a Conrad, W., and Pick, W., Die Herstellung von Hochprozentigem Ferrosilizium im elektrischen Ofen, 1909. Revue de Métallurgie, vol. 9, 1912, p. 362.
^b Guillet, L., Étude industriel des alliages métallique.

con in the product. Using iron turnings, silica, and carbon, he obtained an alloy containing 22 per cent silicon. In 1875 Pourcel made 15 per cent ferrosilicon in the blast furnace at Terre-Noire. Silicospiegel was made at the same time in the blast furnace of Pourcel. From that time on, the manufacture of ferrosilicon in the blast furnace became firmly established. Moissan^a in his electric furnace work made some ferrosilicon and silicides. The first company to take up the manufacture of ferrosilicon in the electric furnace was a company in West Virginia operating under the Chalmot patents^b in 1898. Thus the manufacture of this important alloy in the electric furnace was begun in the United States, but to-day comparatively little is produced in this country. Blast-furnace ferrosilicon had been manufactured for many years previous to this in the United States. In 1899 the carbide industry in Europe began to collapse, and the manufacture of ferro-alloys was commenced. The first product made was ferrosilicon, the production of which was commenced in 1899 at Bozel, France, under the patents of Rothenau.^c This was followed by a plant at Meran, Switzerland, in 1900, a plant at Giffre, Switzerland, in 1901, and plants at Matrei, Austria, and at Notre Dome de Briarçon, Ugine, Livet, and Albertville, France, in 1902. There are to-day many other plants in France, Germany, Switzerland, Austria, Norway, and Sweden. In the United States there is but one producer, a company that operates an old aluminum plant at Kanawha Falls, W. Va., and also one at Niagara Falls, N. Y. It is said that not all of the ferrosilicon sold by this company is manufactured in this country, some being imported. At Welland, Ontario, Canada, there is a ferrosilicon works producing about 4,000 tons yearly. The product contains 50 per cent of silicon. Some of it is exported to the United States.

THEORY.

SILICIDES OF IRON.

A silicide of iron in the proportion 87.28 per cent iron and 11.01 per cent silicon was found in a meteor by Shepard in 1859. Carnot and Goutal^d isolated the silicide $(\text{FeMn})_3\text{Si}$ by treatment of a silicon spiegel with sulphuric acid. This corresponds in silicospiegel to the silicide Fe_3Si , 85.714 per cent iron and 14.286 per cent silicon, that possibly exists in ferrosilicon. Naske^e tried to obtain pure Fe_3Si by treating a ferrosilicon containing 13.19 per cent silicon with ammoniacal copper chloride, but was unsuccessful. By treating a 10 per

^a Moissan, H., Action du silicium sur le fer, le chrome et l'argent: *Compt. Rend.*, vol. 121, 1895 p. 621.

^b U. S. patents Nos. 602975 and 602976, Apr. 26, 1898.

^c Carnot, and Goutal, Recherches sur l'état où se trouvent le silicium et le chrome dans les produits siderurgiques: *Compt. Rend.*, vol. 126, 1898, p. 1240.

^d Carnot, and Goutal, *Loc. cit.*

^e Conrad, W., and Pick, W., Die Herstellung von Hochprozentigem Ferrosilizium in elektrischen Ofen, 1909. *Rev. de Métallurgie*, vol. 9, 1912, p. 362.

cent ferrosilicon with sulphur, he obtained a magnetic product which resembled a mixture of FeS and Fe_3Si . Although the silicide Fe_3Si has not been actually isolated, its existence seems probable by analogy with cementite Fe_3C and $(\text{FeMn})_3\text{C}$. Fe_3Si is decomposed after a long time by dilute sulphuric acid, and is dissolved by hydrochloric acid from all silicides containing less than 20 per cent silicon. It is not attacked by dilute nitric acid.

The silicide Fe_2Si , 80 per cent iron and 20 per cent silicon, has been isolated from commercial products by Carnot and Goutal,^a by Osmond,^b and by Lebeau.^c Carnot treated ferrosilicon with dilute nitric acid and ammoniacal copper chloride and Goutal treated it with warm dilute nitric acid. Hahn obtained Fe_2Si by synthesis, fusing iron chloride, sodium chloride, amorphous silicon, sodium, and fluorspar. In the electric furnace Moissan^d heated iron oxide and crystallized silicon to obtain Fe_2Si . Lebeau obtained it by heating a mixture of iron and cuprosilicon.

Fe_3Si forms small prismatic crystals of metallic appearance. It is magnetic and has a specific weight of 7. It is not attacked by either dilute or concentrated nitric acid. It is dissolved easily by hydrochloric acid, with more difficulty by aqua regia, and completely by hydrofluoric acid. In alkaline solution it is dissolved completely only in hot concentrated solution.

The silicide Fe_3Si_2 , which is found frequently in industrial products bearing 25 to 28 per cent silicon, contains 75 per cent iron and 25 per cent silicon. It forms octahedral crystals and has a specific gravity of 6.7. It is as magnetic as Fe_3Si and Fe_2Si . It is stable in the presence of all acids. Chlorine or bromine attacks it when at a red heat. According to Gin, it is decomposed at a high temperature into Fe_2Si and Si.

The best known silicide of iron is FeSi , 66.67 per cent iron and 33.33 per cent silicon. This alloy was prepared by synthesis by Fremy, through the action of silicon chloride upon iron, and by Lebeau through the action of cuprosilicon on iron. Lebeau also isolated FeSi from ferrosilicon containing 35 per cent silicon. FeSi crystallizes in tetrahedral needles. Its specific weight is 6.17, and its hardness about 7. When cold it is attacked by fluorine; at a dull red heat it is attacked by bromine and chlorine, by an alkaline fusion, by hydrofluoric acid, or by a mixture of hydrofluoric and nitric acids.

The silicide FeSi_2 , 50 per cent each of iron and silicon, has been little studied. It has been prepared by Hahn and Lebeau by syn-

^a Carnot, and Goutal, *Recherches sur l'état où se trouvent le silicium et le chrome dans les produits siderurgiques*: *Comp. Rend.*, vol. 126, 1898, p. 1240.

^b Osmond, F., *Recherches calorimetriques sur l'état du silicium et de l'aluminium dans les fers fondus*: *Compt. Rend.*, vol. 113, 1891, p. 474.

^c Conrad, W., and Pick, W., *Die Herstellung von Hochprozentigem Ferrosilizium im elektrischen Ofen*, 1909. *Rev. de Métallurgie*, vol. 9, 1912, p. 362.

^d Moissan, H., *Action du silicium sur le fer, le chrome et l'argent*: *Compt. Rend.*, vol. 121, 1895, p. 621.

thetic methods. Chalmot treated 39 to 50 per cent ferrosilicon with cold dilute hydrofluoric acid to obtain it. Its specific weight is 5.4 and its hardness 4 to 5. Hydrofluoric attacks it only in concentrated solution, and the other acids have no effect upon it.

The silicide FeSi_3 , 40 per cent iron and 60 per cent silicon, is also little known. Naske obtained it from a ferrosilicon by treatment with hydrofluoric acid. It was in the form of a crystalline powder, which was insoluble in all solvents and attacked with difficulty by carbonate fusion. Pick claims that this silicide exists only in ferrosilicon containing a high percentage of silicon.

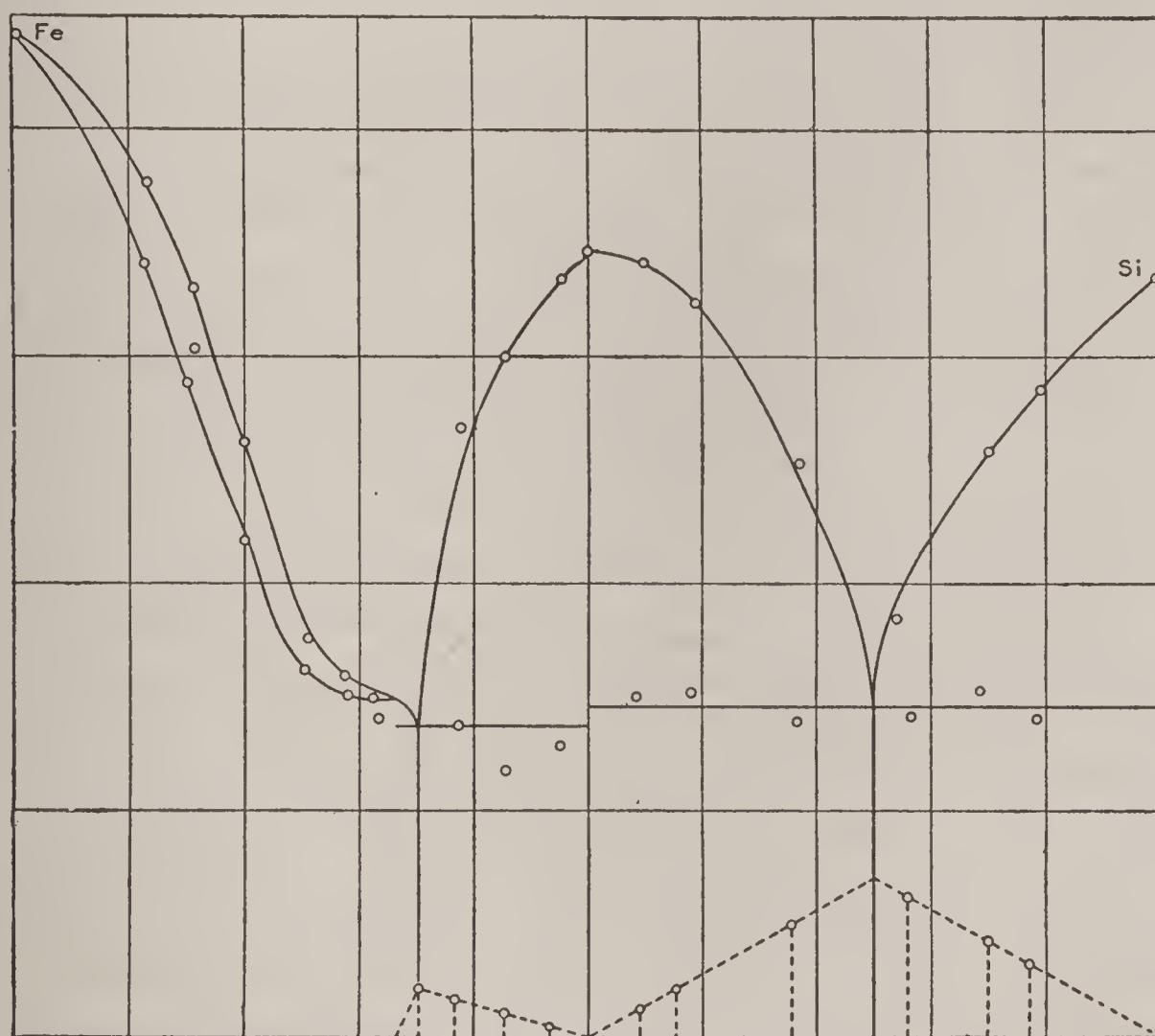


FIGURE 53.—Cooling curves of the iron-silicon system.

Tamman ^a and Gürtler have obtained the cooling curves of the iron-silicon system (fig. 53). They show the existence of only two silicides, Fe_2Si and FeSi , but from work of others it seems that there surely exist three, Fe_2Si , FeSi , and FeSi_2 , and perhaps two others, Fe_3Si and FeSi_3 .

The resistance of silicides of iron to acids increases and the specific weight and magnetic power decrease when the silicon content increases. They all have a good electrical conductivity and are disintegrated by carbonate fusion more or less easily according to their silicon content.

^a Conrad, W., and Pick, W., Die Herstellung von Hochprozentigem Ferrosilizium im elektrischen Ofen, 1909. Rev. de Métallurgie, vol. 9, 1912, p. 362.

PROPERTIES OF FERROSILICON.^a

Ferrosilicon always has a crystalline appearance. If its silicon content is less than 20 per cent it is dull, shows a fine crystallization, and resembles white iron. With a silicon content of 20 to 30 per cent the crystals become more brilliant and show a slaty crystallization. With a silicon content of 50 per cent or more, the crystallization disappears and is replaced by a fine texture resembling more and more silicon. It is not difficult to judge the percentage of silicon by the eye, according to the texture and the color of the fractured surface. As the percentage of silicon increases to more than 50 per cent, the ferrosilicon takes on a bluish color.

The various grades of commercial ferrosilicon are substantially binary alloys. Two compounds, Fe_2Si and FeSi , exist. The microstructure indicates two eutectics, one containing 21.6 per cent silicon (Fe_2Si and FeSi) and the other containing 60 per cent silicon (FeSi and Si). With a silicon content of less than 20 per cent the alloys consist of solid solutions of Fe and Fe_2Si , which are hard, firm masses giving off little or no gas. Alloys with a silicon content of 20 to 21.6 per cent silicon consist of primary crystals of Fe_2Si in a ground of a eutectic composed of Fe_2Si and FeSi . These alloys are more brittle than the lower grades. With a silicon content of 21.6 to 33.3 per cent, the structure shows FeSi surrounded by the eutectic Fe_2Si and FeSi . With a silicon content of 33.3 to 60 per cent there are crystals of FeSi in the eutectic FeSi and Si . With more than 60 per cent of silicon there exist silicon crystals in a field of the eutectic FeSi and Si .

With a silicon content of 30 per cent or less, ferrosilicon is hard and has no tendency to disintegrate spontaneously. Samples containing 30 to 65 per cent silicon show a decided tendency to spontaneous disintegration. They crumble on keeping, and some of them actually fall to powder after a few weeks. This spontaneous disintegration is usually accompanied by the evolution of bad-smelling and poisonous gases. Although samples with a silicon content of 70 to 96 per cent are somewhat brittle, they are not so easily broken and reduced to powder as those containing between 30 and 65 per cent silicon, and they do not show a tendency to disintegrate spontaneously.

The specific gravity decreases as the percentage of silicon increases, because the specific gravity of pure iron is 7.8, whereas that of pure silicon is 2.49. In figure 54 a curve representing the results of determinations of the specific gravity of various samples of commercial ferrosilicon, made by Dr. Hahe, of the British Local Government Board, is shown in comparison with the theoretical curve representing the calculated specific gravities for all percentages on the assumption

^a Copeman, S. M., Bennett, S. R., and Hahe, H. W., Yellow book of British Local Government Board on the Manufacture, Uses, and Transport of Ferrosilicon, 1910: Met. and Chem. Eng., vol. 8, 1910, p. 133.

that mixtures of iron and silicon are formed without change of volume. A comparison of the two curves shows that the assumption is not correct, that the specific gravities are partly above and partly below the theoretical curve, and that contraction of volume occurs in the alloys containing 60 per cent or less of silicon, and expansion of volume in alloys containing more than 60 but less than 96 per cent silicon, presumably owing to the presence of definite silicides of iron in the alloys.

According to the diagram of Tamman and Görtler (fig. 53), the fusing point of 25 and 50 per cent ferrosilicon is $1,360^{\circ}\text{C}$. The melting point of silica is $1,425^{\circ}\text{C}$., and of iron $1,540^{\circ}\text{C}$.

The chemical purity of the product decreases slightly with decrease in the silicon content. Compared with the impurities of the

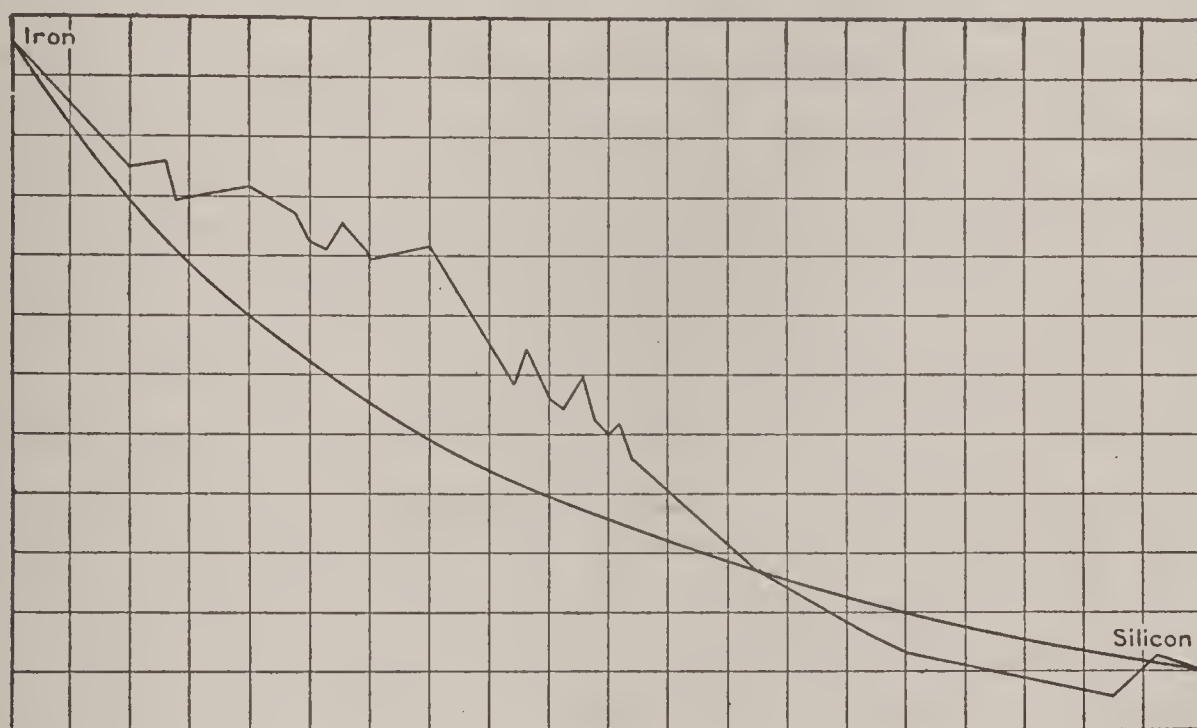


FIGURE 54.—Specific gravity curves of ferrosilicon samples containing varying percentages of ferrosilicon. Irregular line represents results of actual determinations. Uniform curve represents calculations based on assumption that mixtures of iron and silicon are formed without change of volume.

blast-furnace alloys, the impurities in electric-furnace ferrosilicon are low. The calcium content in the 25 per cent grade is about 0.5 per cent, and increases to 0.7 to 1 per cent for 90 per cent ferrosilicon. The magnesium content varies from 0.10 per cent in the 25 per cent grade to 0.3 per cent in the 90 per cent grade. The average aluminum content in all grades is between 0.1 and 0.15 per cent. In the blast-furnace ferrosilicon the percentage of carbon is 1 to 1.5 per cent. This diminishes with increase in the silicon content, so that in the 25 per cent grade there is about 0.35 per cent carbon, and in the 90 per cent grade the carbon content is as low as 0.2 per cent. The phosphorus content is generally less than 0.10 per cent, the sulphur content less than 0.05 per cent, and the manganese content less than 0.4 per cent.

DISINTEGRATION AND EVOLUTION OF GAS FROM FERROSILICON.

As has been stated, a ferrosilicon containing between 30 and 65 per cent silicon has a tendency to disintegrate. It is not known whether this property is inherent in these grades, or whether it is due to chemical or physical conditions. The main impurity that might cause disintegration is calcium. If the calcium were present in the state of a silicide, the alloy would be stable, but if the calcium were present as calcium carbide, the alloy would be very unstable. The presence in ferrosilicon of calcium as calcium carbide might be due to the manufacture of ferrosilicon in an old carbide furnace. In a new furnace there should be no contamination by carbide.

It has been noticed that disintegration occurs in spots. In a mass of ferrosilicon of uniform composition, part will stay solid and part break up. Coating the product with paraffin has been tried, but is no longer believed to be efficacious, as it affects only the surface. Some persons believe that disintegration is affected by the temperature of pouring and the density of the current of electricity in the electrodes. When the temperature is high and the amount of current great, the product seems to be more subject to disintegration. It is possible that there is a phenomenon analogous to the allotropic changes of certain elements, such as sulphur. The solid parts have a homogeneous structure and grain, whereas the substance affected by disintegration consists of crystals surrounded by a powder.

There is evolution of explosive and poisonous gases from these grades of ferrosilicon when the alloy is in moist air and when there is friction between the particles of the alloy as in transportation. Dupre and Lloyd^a attribute the explosions to acetylene gas made to ignite by the presence of small quantities of phosphide of hydrogen (PH_3), and when this is not present by sparks from attrition of particles of ferrosilicon. Wilson^a found silicide of hydrogen. Ashenary^a believed the gas to be hydrogen. Most investigators believe the poisonous gas to be PH_3 , as phosphide of calcium and phosphide of iron are not decomposed by water.

In his report to the British local government board, Hahe states that he found phosphoretted hydrogen (PH_3) and arseniuretted hydrogen (AsH_3). He did not find in the samples tested any siliuretted hydrogen (SiH_4) nor acetylene (C_2H_2), although others report those gases present as in ferrosilicon. It is probable that the poisonous emanations are phosphoretted hydrogen and arseniuretted hydrogen evolved by the action of water.

The formation of these gases is due to the presence of small quantities of calcium phosphate [$\text{Ca}_3(\text{PO}_4)_2$] and arsenic in the raw mate-

^a Conrad, W., and Pick, W., Die Herstellung von Hochprozentigem Ferrosilizium im elektrischen Ofen, 1909. Rev. de Métallurgie, vol. 9, 1912, p. 362.

rials. Calcium phosphate occurs in coal and quartz. In itself it is harmless. It is insoluble in water and is common in nature. In the electric furnace, in the presence of carbon, it is reduced to calcium phosphide (Ca_3P_2). The calcium phosphide remains in the ferrosilicon and in contact with water in moist air is decomposed with evolution of phosphoretted hydrogen (PH_3), which is very poisonous. Arsenic occurs in nature closely associated with phosphorus, and finds its way into the ferrosilicon as calcium arsenide, which is decomposed by water or moist air, with the formation of arseniuretted hydrogen (AsH_3), also a poisonous gas.

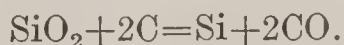
These gases are evolved only by the grades of ferrosilicon that disintegrate easily—containing 30 to 65 per cent silicon. The British Government board classed as dangerous all grades that contained between 30 and 70 per cent silicon.

REDUCTION WITH CARBON.

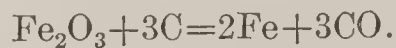
Ferrosilicon is made commercially in two ways only—by the reduction of silica and iron ore with carbon or by the reduction of silica with carbon, the iron being added as iron turnings, which are simply melted. The utilization of siliceous and ferruginous slags has been proposed, but has not been adopted commercially, and experiments of the writer indicate that it does not seem practicable.

Silica is reduced by solid carbon at a temperature of $1,485^\circ \text{C}$. The reduction point of silicon is lowered if there is iron present in the charge to combine with the silica.^a With a charge composed of 20 parts of silica, 30 parts of iron, and 8 parts of carbon the temperature of reduction was lowered to $1,200^\circ \text{C}$. in experiments performed by Greenwood. In commercial practice the presence of iron to lower the reduction temperature is important, as is shown by the fact that furnaces working upon a low-silicon product, with a large charge of iron operate better than when treating a high-silica product with a low iron charge.

Reduction takes place according to the formula:



Thus, the reduction of 100 parts of silicon from 214 parts of silica requires 86 parts of carbon. The reduction of iron oxide is generally by the solid carbon only, according to reaction:



By this formula for the reduction of 100 parts of iron from 143 parts of hematite 32 parts of carbon is necessary. In the usual

^a Greenwood, H. C., Sluele, R. E., Prinz, J. N., Reduction of refractory oxides, production of ferro-alloys and formation of carbides. Trans. Chem. Soc. (England), vol. 93, 1908, p. 1484. Electrochem. and Met. Ind., vol. 7, 1909, p. 119.

ferrosilicon furnace there is no shaft, so that reduction results mainly in the formation of gas, the greater part of which is carbon monoxide, as shown in the above formula.

In the calculation of theoretical power consumption the following constants are used: ^a

Reduction or oxidation.	Calories absorbed.
1 kg. Fe from Fe ₂ O ₃	1, 746
1 kg. Si from SiO ₂	6, 428
1 kg. C to CO.....	2, 430

The specific heat of iron for temperatures ranging from 0° to 1,400° C. is taken as the average 0.20; of silicon for temperatures ranging from 0° to 1,400° C., 0.25; and of carbon for a similar range, 0.50.

Theoretically the energy necessary for the reduction of 1 kg. (2.2 pounds) of silicon from silica is the result of dividing 6,428 by 857, or 7.5 kilowatt-hours per kilogram (3.4 kilowatt-hours per pound). The amount of energy necessary to reduce a long ton of pure silicon would then be 7,600 kilowatt-hours, or 0.87 kilowatt-year, per ton.

PRODUCTION FROM IRON ORE OF FERROSILICON CONTAINING 30 PER CENT SILICON.

In the calculations following, relative to producing ferrosilicon containing 30 per cent silicon from iron ore, the assumptions made are: First, that the iron of the alloy is to be reduced from a pure hematite ore 70 per cent iron; second, that the ferrosilicon is to contain 30 per cent iron and 70 per cent silicon; third, that the silicon is to be from a pure quartz containing 46.7 per cent silicon; and, fourth, that the reduction is to take place according to the given reactions. The calculations follow:

Reduction.	Calories required.
700 kg. Fe from Fe ₂ O ₃700×1,746.....	1, 222, 200
300 kg. Si from SiO ₂300×6,428.....	1, 928, 400
Heating.	
700 kg. Fe to 1,400° C.....700×1,400×0.2.....	196, 000
300 kg. Si to 1,400° C.....300×1,400×0.25....	105, 000
482 kg. C to 1,400° C.....482×1,400×0.5.....	337, 400
Total.....	3, 789, 000
Amount of heat supplied:	
By combustion of 482 kg. C.....482×2,430....	1, 171, 260
By the electric current.....	2, 617, 740
Total.....	3, 789, 000

Therefore, the theoretical amount of electrical energy necessary for the production of 30 per cent ferrosilicon from iron ore and quartz is the result of dividing 2,617,740 by 857, or 3,054 kilowatt-hours per metric ton, or, in terms of the long ton, 3,109 kilowatt-hours, or 0.35 kilowatt-year per long ton.

^a Richards, J. W., Metallurgical calculations, 1908, vol. 1.

PRODUCTION OF FERROSILICON CONTAINING 30 PER CENT SILICON FROM IRON TURNINGS.

In the calculations following, relative to producing ferrosilicon containing 30 per cent silicon from iron turnings, the assumptions made in the ore are as follows, the same as for the production of 30 per cent ferrosilicon using iron ore, except that in this case wrought iron or steel turnings are the source of iron:

Reduction.		Calories required.
300 kg. Si from SiO_2	$300 \times 6,428$	1, 928, 400
Heating.		
700 kg. Fe to $1,400^\circ \text{C}$	$700 \times 1,400 \times 0.2$	196, 000
300 kg. Si to $1,400^\circ \text{C}$	$300 \times 1,400 \times 0.25$	105, 000
258 kg. C to $1,400^\circ \text{C}$	$258 \times 1,400 \times 0.5$	180, 600
Total.....		2, 410, 000
Amount of heat supplied:		
By combustion of 258 kg. C	$258 \times 2,430$	626, 940
By the electric current.....		1, 783, 060
Total.....		2, 410, 000

Therefore the theoretical amount of electrical energy necessary for the production of 30 per cent ferrosilicon from iron turnings and quartz is the result of dividing 1,783,060 by 857, or 2,080 kilowatt-hours per metric ton, or 2,116 kilowatt-hours (0.24 kilowatt-year) per long ton.

From these figures it is evident that the production of 30 per cent ferrosilicon in the electric furnace when iron turnings are used requires only 68 per cent of the energy necessary with the use of iron ore.

FIFTY PER CENT FERROSILICON.

Analogous to the case of a 30 per cent ferrosilicon, with a 50 per cent product the power consumption with iron ore is 3,945 kilowatt-hours per metric ton, or 4,010 kilowatt-hours (0.46 kilowatt-year) per long ton. If iron turnings are used instead of iron ore the power consumption per ton of 50 per cent ferrosilicon made is 3,245 kilowatt-hours per metric ton, or 3,300 kilowatt-hours (0.38 kilowatt year) per long ton.

In making 50 per cent ferrosilicon, the saving in energy is not so great by using iron turnings instead of iron ore as in making the 30 per cent product because of the smaller amount of iron necessary in the charge. In making the 50 per cent alloy the energy consumption with turnings is 82 per cent of that with iron ore.

MANUFACTURE OF FERROSILICON IN THE BLAST FURNACE.

Ferrosilicon of less than 20 per cent silicon is made in the blast furnace, whereas the electric furnace is used for all the grades containing a higher percentage. The grades made in the blast furnace

contain 10 to 20 per cent silicon, most of them containing less than 15 per cent silicon. Higher grades can not be produced in the blast furnace because of the impossibility of obtaining a high enough temperature. The wear on the furnace lining is great, and in order to maintain the high temperature necessary, a large quantity of coke is employed. As in the electric furnace, the silicon is reduced by solid carbon. Reduction is said to be more easy if the slag is low in lime and high in alumina—probably because the silicon does not have as strong an affinity for alumina as for lime. The charge used in the blast furnace may consist of siliceous iron ore, hammer scale, and coke, a hot blast being used. Gin gives as a charge: turnings 1,000 parts, quartz 410 parts, and coke 940 parts.^a

In small furnaces the production is 30 to 40 tons per day, and with the large furnaces 80 to 100 tons per day. The blast-furnace operation of making ferrosilicon is similar to making pig iron, except that, owing to the highly siliceous slags, the steady operation of the furnace is more difficult. A small amount of blast-furnace ferrosilicon is imported, but most of it is produced here.

MANUFACTURE OF FERROSILICON IN THE ELECTRIC FURNACE.

RAW MATERIALS USED.

There are two general kinds of charge used in the manufacture of ferrosilicon in the electric furnace. The charge may consist of wrought iron, cast iron or steel turnings, quartz or sand, and charcoal, coal, or coke; or siliceous iron ore, quartz or sand, and charcoal, coal, or coke may be used. The choice of the source of iron depends chiefly on the material available. Iron in the form of turnings of some sort is more commonly used than iron ore, because, as has already been mentioned, it consumes much less power—68 per cent of the amount used for a 30 per cent alloy with iron ore, and 82 per cent for the 50 per cent product. The operation of a ferrosilicon furnace operating on turnings is much steadier and requires less attention because there is not as much slag formed as in the ore process. The purer the raw materials the purer is the product obtained. For this reason wrought iron or steel turnings are preferable to cast-iron borings and are used by most works. At the Girod plant, at Ugine, the average composition of the iron turnings used is 0.4 to 0.5 per cent carbon, 0.15 to 0.25 per cent silicon, 0.5 to 0.7 per cent manganese, 0.6 to 0.9 per cent sulphur, and 0.08 to 0.10 per cent phosphorus. Cast-iron turnings contain as high as 0.5 to 0.8 per cent phosphorus. At works at Bozel, France, steel or wrought-iron turnings containing less than 0.3 per cent phosphorus are used. Another French company, at Livet, uses steel turnings containing less than 0.10 per cent phosphorus.

^a Conrad, W., and Pick, W., Die Herstellung von Hochprozentigem Ferrosilizium im elektrischen Ofen, 1909. Rev. de Métallurgie, vol. 9, 1912 p. 352.

The St. Marcel, France, works of the Société d'Industrie Electro-Chimiques La Voltes use steel shavings for 30 per cent ferrosilicon, and iron ore 95 per cent Fe_2O_3 for the 50 per cent grade. At Kanawha Falls, W. Va., iron ore is used in the production of 30 per cent ferrosilicon.^a The ore contains 62.34 per cent iron, 7.93 per cent SiO_2 , 0.09 per cent MgO , 0.72 per cent CaO , 0.4 per cent phosphorus, and a trace of sulphur. Gin in some experiments used a highly siliceous^b iron ore containing 59.2 per cent Fe_2O_3 , 2.1 per cent MnO , 24.6 per cent SiO_2 in producing a 50 per cent ferrosilicon.^b

For the supplying of silicon in the charge, quartzite is preferable to sand because it is more nearly pure, and hence does not cause the formation of as much slag, which clogs the furnace. Also, the fine state of the sand seems to cause less regular operation of the furnace. The quartzite is usually crushed to about 2-inch size. At the Bozel plant mentioned above, the quartzite charged contains 95 per cent SiO_2 , 4 per cent Al_2O_3 , and traces of calcium and phosphorus. The Girod works use quartzite containing 92 per cent SiO_2 and traces of phosphorus and sulphur. At St. Marcel the quartzite has 98 per cent SiO_2 , and a trace of MgO .

Whether charcoal, coke, or coal is used depends largely upon the cost of those materials in the district where a given plant is located. As the materials are used only as reducing agents and as, because of the absence of a shaft, no strength is necessary to support the charge in the ferrosilicon furnaces, the cheapest reducing material available is used. In France the works are all situated in the mountain districts where transportation costs are high, so that the local anthracite coal is largely employed. Most of such coal is rather high in ash, and when grades of ferrosilicon containing 75 per cent or more silicon are being manufactured it is customary to employ charcoal to avoid slag in the furnace. The slag increases the difficulties of operation considerably in the production of high-grade ferrosilicon and also increases the furnace impurities that would pass into the product. At the Girod plant in Ugine, coal guaranteed not to average more than 8 per cent ash or more than 0.005 per cent phosphorus or sulphur is used. The St. Marcel works employ gas coke for the production of 50 per cent grades, and a cheaper anthracite coal for the 25 per cent alloy or other low grades. A French company at Livet uses anthracite coal containing 7 per cent ash, 0.013 per cent phosphorus, 0.41 per cent sulphur, and no arsenic. At Bozel for the lower percentage alloys—50 per cent or less—anthracite coal containing 20 per cent ash is charged, but in the manufacture of the 80 per cent grade, charcoal is employed. In Norway and Sweden, where most of the ferrosilicon works are on tidewater or on a canal, anthracite coal of a high grade is generally used.

^a Schoel, G. P., *Manufacture of ferro-alloys in the electric furnace: Electrochem. Ind.*, vol. 2, 1904, p. 396.

^b Schoel, G. P., *loc cit.*

In almost all plants the purest raw materials available are obtained for making ferrosilicon. As already mentioned, the introduction of slag-forming materials into the charge is thus prevented, and the presence of objectionable impurities such as phosphorus and sulphur in the product is avoided. With raw materials containing bases the silica intended for the product combines with the bases to form slag with such a high percentage of silica as to be very sticky, so that the pieces can be removed from the furnace only by fishing them out as they work to the surface around the electrode, although some of the slag may be tapped with the ferrosilicon. Owing to the strongly reducing nature of the process most of the phosphorus and sulphur charged goes into the metal. The sulphur, however, is slagged to a small extent.

CHARACTER OF FERROSILICON PLANTS.

Ferrosilicon plants are arranged and use the same general types of furnaces as have already been described. The Meraker, Ugine, and Livet works may be considered as typical. Conrad and Pick ^a have made a design of a works using 2,000 kilowatts, as shown in figures 55 and 56. There are two three-phase furnaces, each of 2,000 kilowatts capacity. Only one is operated at a time, the others being held in reserve. Each furnace has its own three transformers connected each to a phase in delta connection. The quartz is hauled up an incline to a crusher above, from which it falls into bins below. From the bins it is drawn into a car mixed with turnings and coal and raised on an elevator to the charging floor of the furnace. The general arrangement of the works is similar to that of other works. The total space occupied by the plant is 162,000 square feet distributed as follows:

	Square feet.
Electric furnaces.....	12, 950
Transformers.....	6, 470
Packing room.....	4, 850
Setting up of electrodes.....	3, 240
Crushing and mixing.....	6, 470
Repair room.....	2, 580
Coal room.....	12, 950
Turnings store.....	3, 240
Quartz store.....	32, 400
Finished product room.....	6, 470
Tool room.....	3, 240
Office, laboratory.....	4, 850
Yard, tracks.....	64, 800
	<hr/> 162, 000

^a Conrad, W., and Pick, W., Die Herstellung von Hochprozentigen Ferrosilizium im elektrischen Ofen, 1909. Rev. de Métallurgie, vol. 9, 1912, p. 362.

PRACTICE.

METHOD OF CHARGING.

The raw materials are all thoroughly mixed before being charged into a ferrosilicon furnace. Charging may be from the tapping floor by a man shoveling the charge into the top of the furnace, or charg-

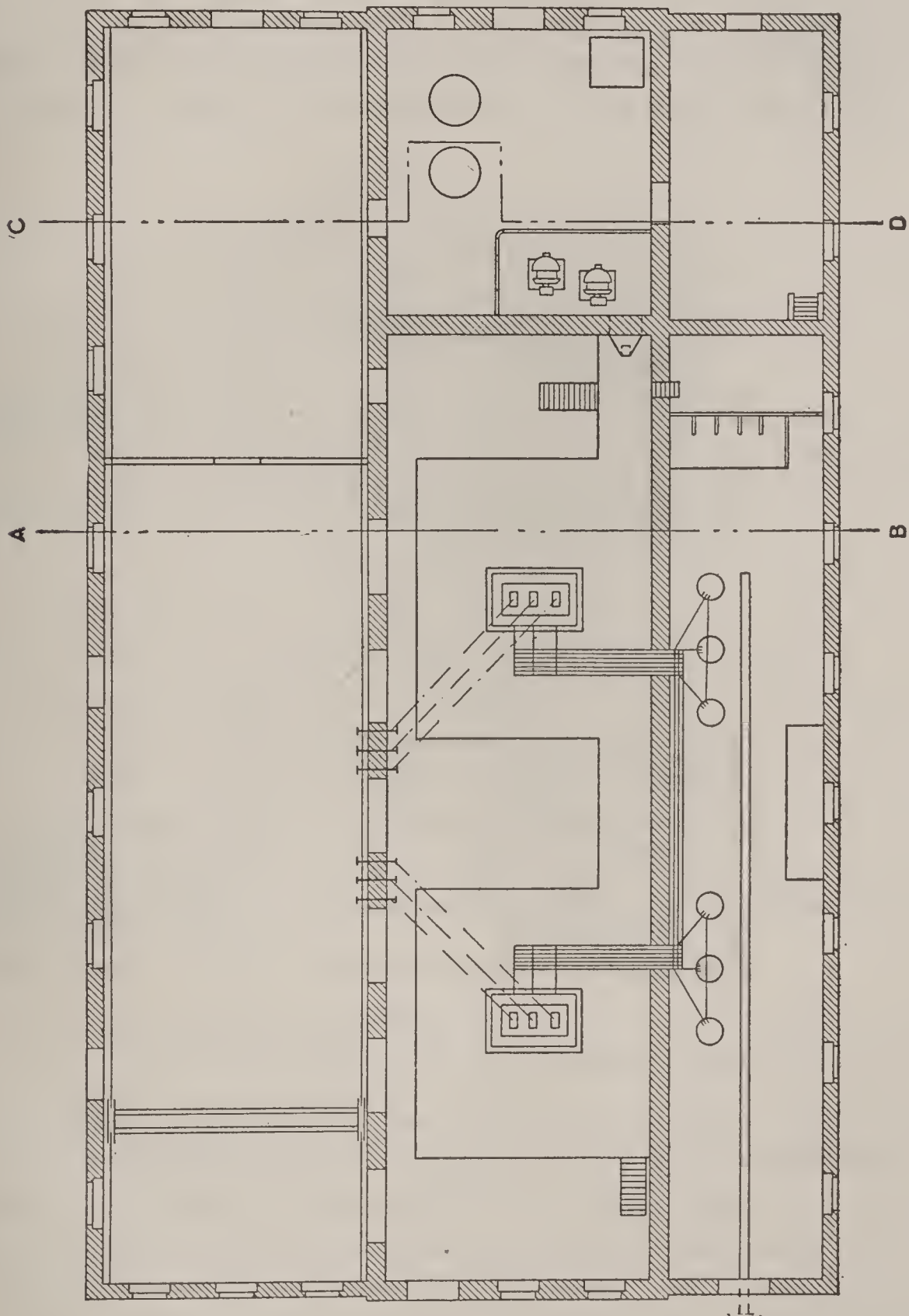


FIGURE 55.—Plan of ferrosilicon plant designed by Conrad and Pick.

ing may be from a floor at the furnace top, the usual arrangement in modern works. In charging the material is placed at intervals around the electrodes, so that the top of the furnace never contains molten material except when some works up around the electrode. The electrode projects about 1 foot into the charge, and around the

end the material is molten and maintains a space of a fraction of an inch, so that the arc is maintained in the gas formed in the space. The arc is called a freeburning arc. By this method there is a lower power consumption than if the furnace is operated as a resistance furnace entirely. It is essential that the top of the furnace be covered with unmelted material, so as to cause less electrode and raw-material consumption. By operation with a covered bed of fusion, the silicon which is volatilized at a temperature higher than $2,000^{\circ}$ C. is condensed in the upper part of the furnace. The unmelted charge also causes the coal consumption to be lower. In producing

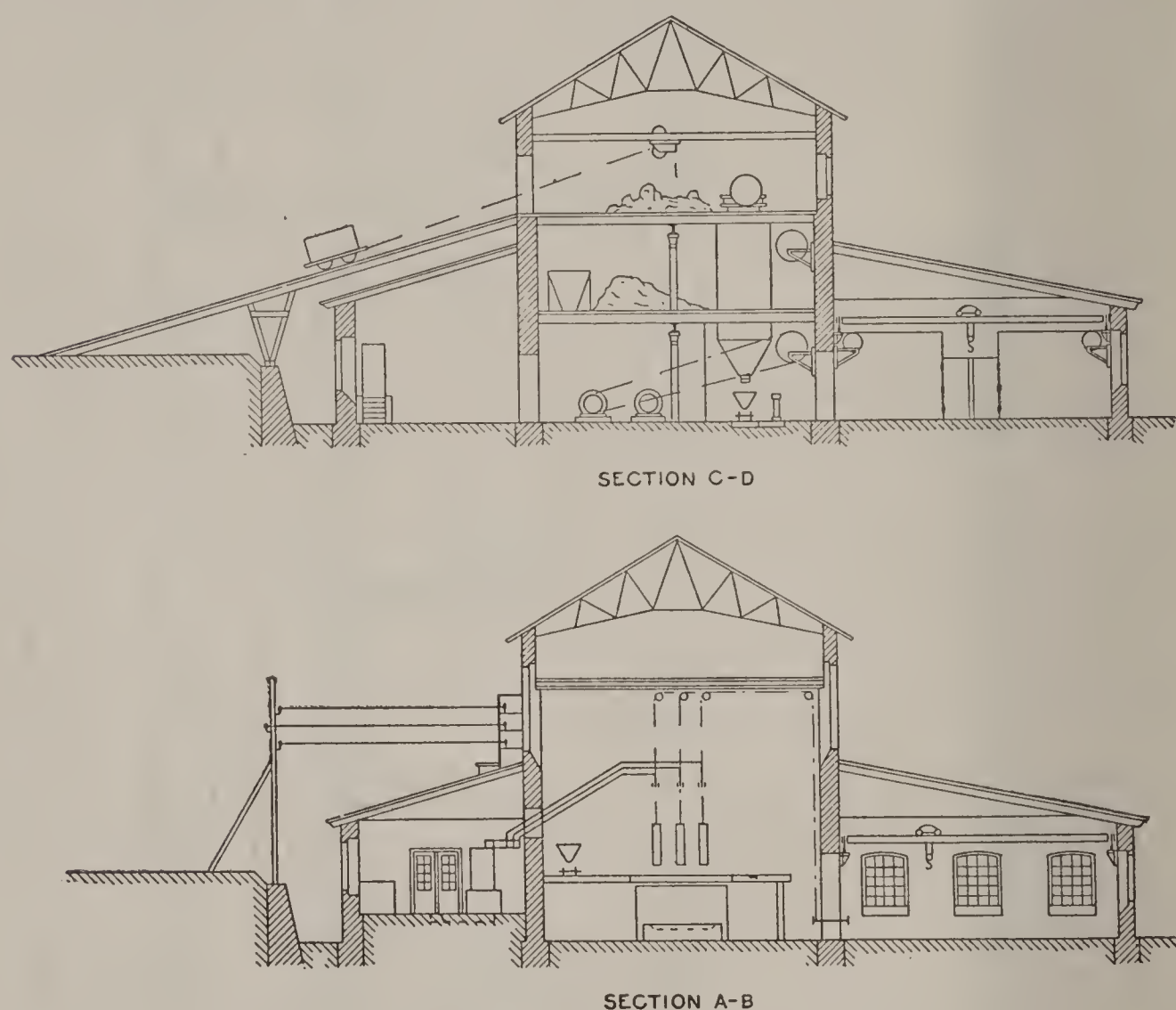


FIGURE 56.—Elevation of Conrad and Pick's ferrosilicon plant.

50 per cent ferrosilicon in a furnace operated with an uncovered bed there were losses of 53.7 per cent of the quartz charged and 43.5 per cent coal by volatilization and oxidation,^a as compared with 13.01 per cent quartz and 27.58 per cent coal in a modern furnace working with a cold top. As a result of this decrease of losses the efficiency is much increased. In the furnace operating with an uncovered bed and making 50 per cent alloy the power consumption was 7.25 kilowatt-hours per pound, or 16,350 kilowatt-hours per long ton. In operation with a covered bed the energy used was 3.16 kilowatt-hours per pound,

^a Conrad, W., and Pick, W., Die Herstellung von Hochprozentigen Ferrosilizium im elektrischen Ofen, 1909. Rev. de Métallurgie, vol. 9, 1912, p. 362.

or 7,100 kilowatt-hours per long ton. With the furnace having an uncovered bed the efficiency was 25 per cent, and with the furnace having a covered bed the efficiency was 57.28 per cent. The low efficiency of the furnace with an uncovered bed was not due entirely to the system of operation, but was also due to the fact that the furnace was of 200 kilowatts capacity, as compared with a capacity of 1,500 kilowatts for the furnace with the covered bed. These examples also show the great strides made in the improvement of furnaces and of operating efficiency in the 10 years from 1900 to 1910, as the small furnace was one of the first to be built and the larger one of modern design.

TAPPING.

The furnace is tapped as seldom as possible. The necessity of tapping is indicated by the irregularity of the current as shown by the meters and the irregular blowing of gas out of the furnace, also, when the metal has been too close to the electrode there is an intermittent pounding sound. The shortest interval is about two hours. The length of the intervals depends upon the percentage of silicon in the product and the size of the furnace. In tapping, an iron rod is driven into the tap hole. If material with a high percentage of silicon is being made, it is sometimes necessary to attach the rod to the circuit and burn out the hole with an arc. The tap hole is plugged with fine clay or some similar refractory material.

A furnace operating steadily in making ferrosilicon has white fumes of silicon passing off at the top with carbon monoxide burning to dioxide around the electrodes. In some laboratory experiments by the writer, the condensed fumes from a ferrosilicon furnace contained 67.69 per cent SiO_2 , 9.72 per cent Al_2O_3 , 4.11 per cent FeO , 0.20 per cent MgO , and 1.90 per cent CaO , the rest probably being carbon.

CHANGING ELECTRODES.

When it is necessary to shut off the current in changing an electrode, if the shutdown be a long one, bars of iron are stuck down through the charge to make contact with the bottom or between the electrodes, so as to avoid difficulty in starting up. The molten charge is a good conductor, but is not so when solid.

Electrodes are replaced by simply swinging one in over the furnace to replace the holder and the burnt electrode removed. If the electrode has a tendency to rise in the furnace when a fixed voltage is being maintained by hand regulation of the electrode, quartz, a relatively poor conductor, is charged around the electrode, so that it becomes necessary to lower the electrode to maintain the current steadily. If the reverse is the case, coal is added to put in a good conductor for raising it.

SLAG FORMATION.

Because of the use of pure raw materials, comparatively little slag is formed in a ferrosilicon furnace. The slag formed consists of silicates of aluminum and lime or magnesia. They are very high in silica and rather infusible. The addition of more basic materials to make them fusible would do no good, because it would simply draw more silica from the charge and increase the total slag. As the percentage of silicon in the product increases, the specific gravity of the product decreases, so that it is difficult to obtain separation of slag and metal if the slag is sticky. In the manufacture of the 25 per cent alloy, slag is permissible, with a silicon content of 50 per cent, operation is difficult and with higher percentages is impossible if slag is present. As a rule, these slags are so thick that they are not tapped but boil up around the electrode and are removed there. The results of some analyses ^a are as follows:

	Per cent.	Per cent.	Per cent.
SiO ₂	68.86	67.60	79.25
Al ₂ O ₃	13.80	19.63	9.04
CaO.....	26.40	17.00	19.17
Fe ₂ O ₃	1.96	1.02	3.50

The ferrosilicon is tapped into cars lined with sand or carbon. It is dumped to cool and broken into small pieces with hammers and weights. In tapping, iron must be kept away from the molten ferrosilicon as in the molten condition it dissolves iron readily. The product is usually packed in 100-kg. (220-pound) iron drums or in kegs.

RESULTS OF EARLY OPERATIONS.

In the early operations there was much excess raw material charged for the amount of product produced. In a small 200-kw. furnace of the old design, the following charges were made:^b Twenty-five per cent product—40 parts quartz, 40 parts iron, 20 parts of anthracite; 50 per cent product—58 parts quartz, 13 parts iron, 29 parts anthracite; 70 per cent product—66 parts quartz, and 34 parts anthracite. The coal contained 20.71 per cent ash (2 to 3 per cent SiO₂), 71.29 per cent carbon, 1.08 per cent hydrogen, 1.11 per cent water, and 0.48 per cent sulphur. The turnings had 98 per cent iron. The quartz was 98.5 per cent SiO₂, and 0.5 per cent Fe₂O₃. For a 50 per cent ferrosilicon the following charge was used in an 1,800-kw. modern furnace and may be considered as representative practice: Fifty-four parts of quartz, 20 parts of iron turnings, 30 parts of charcoal. The quartz contained 95 per cent SiO₂ and 3.9 per cent Fe₂O₃; the turnings, 98.5 per cent iron; and the charcoal, 71.9 per cent carbon and 4 per cent ash.

^a Conrad, W., and Pick, W., Die Herstellung von Hochprozentigen Ferrosilizium in elektrischen Ofen, 1909. Rev. de Métallurgie, vol. 9, 1912, p. 362.

^b Conrad, W., and Pick, W., Loc. cit.

POWER CONSUMPTION.

The power consumption with this charge was 7,100 kilowatt-hours per long ton of 50 per cent ferrosilicon. Of the total energy 57.28 per cent was usefully used, 8.61 per cent was used in volatilizing silicon, and 0.23 per cent in volatilizing iron, and 33.88 per cent was lost by induction, radiation, and other losses. The electrode consumption during a month of steady operation averaged 84 pounds per long ton of 50 per cent product. Amorphous-carbon electrodes were used.

When iron turnings are used, the power consumption of a modern furnace of 750 kilowatts or more capacity is 6,800 to 8,000 kilowatt-hours per ton of 50 per cent ferrosilicon, and about 3,500 to 4,000 kilowatt-hours for the 25 to 30 per cent grade. As has already been noted in the discussion of the theory of power consumption, the power consumption is considerably higher if iron ore is used instead of turnings. With the ore mentioned on page 165, one company made 28 per cent ferrosilicon with a power consumption of 5,930 kilowatt-hours per ton.^a

Louis^b estimates the power consumption in the production of various grades of ferrosilicon, as shown in the table below, the figures being based on a 7,500-kw. plant and 750-kw. units:

Power consumption in ferrosilicon manufacture.

Raw materials.	Silicon in product.	Power consumption per pound.	Power consumption.	
	Per cent.	Kilowatt-hours.	Kilowatt-hours.	Kilowatt-years.
Quartzite, coke, iron turnings.....	25	1.82	4,080	0.46
Do.....	50	3.64	8,160	.92
Do.....	75	5.46	12,240	1.38
Slag and coke.....	25	1.82	4,080	.46
Slag, charcoal, sand.....	30	2.18	4,890	.56
Do.....	35	2.54	5,600	.64
Do.....	50	3.64	8,160	.92
Quartz, iron ore, coke, or anthracite.....	25	2.18	4,890	.56
Do.....	50	4.36	9,780	1.12
Enrichment of 25 per cent.....	50	4.36	9,780	1.12

WORKMEN REQUIRED.

A small furnace of 200 to 300 kw. requires one man for charging. Two men for six furnaces can perform the remaining duties, such as tapping and mixing charges, the total number necessary for six furnaces, being eight men. A large furnace of 2,000 to 3,000 kw. can be entirely operated by five men.

^a Schoel, G. P., Manufacture of ferro-alloys in the electric furnace: *Electrochem. Ind.*, vol. 2, 1904, p. 396.
^b Louis, J., La fabrication des ferrosilicium au four electrique: *Jour. du Four Electrique et de Lumière*, Oct. 1, 1910, p. 415.

PRODUCTS.

Electric-furnace ferrosilicon is produced in four grades, classified according to silicon content, which is as follows for each grade:

Twenty-five to thirty per cent, 45 to 50 per cent, 75 to 80 per cent, and 90 to 95 per cent. The selling of these products is based entirely upon the silicon content, because the carbon content, owing to its usual low percentage, has no effect upon the price, as in other alloys. The current prices for the various grades are given in the table on page 140. Below are given the results of analyses of the four grades of electric-furnace ferrosilicon:

Results of analyses of four grades of ferrosilicon.^a

Constituent.	Silicon content.			
	25 to 30 per cent.	45 to 50 per cent.	75 to 80 per cent.	90 to 95 per cent.
Silicon.....	30.50	49.50	78.00	88.50
Iron.....	68.00	49.00	20.00	9.30
Manganese.....	.35	.30	.25	.15
Aluminum.....	.10	.15	.10	.15
Magnesium.....	.10	.15	.30	.30
Carbon.....	.35	.30	.30	.25
Sulphur.....	.02	.015	.015	.015
Phosphorus.....	.04	.03	.03	.03

^a Girod, P., Studies on the electrometallurgy of ferro-alloys and steels: Trans. Faraday Soc., vol. 6, 1911, p. 172.

In spite of the agitation against the manufacture of ferrosilicon containing between 38 and 65 per cent silicon because of the poisonous gases that are claimed to be given off, this grade is the one most largely made. Moreover, many persons believe that the gases can be eliminated if the ferrosilicon is made from pure materials.

PACKING AND TRANSPORTATION OF FERROSILICON.

Details on the packing of ferrosilicon in Sweden have recently been obtained by the Department of Labor.^a The Gullspangs Elektrokemiska Aktiebalag use iron-bound wooden cases in two sizes, one for the 50 per cent. product containing 130 kilograms (287 pounds), and one for the 75 per cent product containing 160 kilograms (353 pounds). The cases are unlined but are made of rabbeted boards. Recently the practice of packing the ferrosilicon in sheet-metal cylinders or drums containing 170 kilograms (375 pounds) was begun. The drums have four holes of about ¼-inch diameter in the top to prevent excessive gas pressure.

The Aktiebolaget Heroult's Elektriska Stal until recently packed ferrosilicon in old wooden oil barrels. Now the company has adopted

^a Anon., Packing and transport of ferrosilicon in Sweden: Met. and Chem. Eng., vol. 11, 1913, p. 87.

sheet-iron barrels with wooden heads. The barrels hold 300 kilograms (660 pounds).

The regulations for transportation differ widely. In Germany water-tight packing is required. As a result, ferrosilicon is packed in barrels that are both water and gas tight. Some English railroads require that each package must have three 1-inch holes to permit the escape of gases. For safe transportation uniform regulations should be enforced by all countries.

COST OF MANUFACTURE.

The cost of manufacture of ferrosilicon depends more upon the price of electric power than upon any other item. This is because of the large amount of power necessary and the low cost of the raw materials composing the charge. Consequently a large part of the electric-furnace ferrosilicon used in the United States is imported from Europe and Canada. Of plants engaged in the manufacture of ferrosilicon the writer found none in which the power cost was over \$20 per kilowatt-year and the average was about \$13 per kilowatt-year.

An estimate of the cost of production of 50 per cent ferrosilicon is given in the table below. The calculations are on the basis of two 2,000-kilowatt furnaces, one operating and the other kept in reserve. The Norwegian plant is within 60 miles, the French plant within 200 miles, and the American works assumed to be within 100 miles of sea-board. Labor conditions are considered to be as found in the various countries.

Cost of production of 50 per cent ferrosilicon per ton (2,240 pounds).

Item.	Norway.		France.		United States.	
	Cost of unit.	Total cost.	Cost of unit.	Total cost.	Cost of unit.	Total cost.
1,170 pounds of iron.....	\$14.00	\$7.30	\$14.00	\$7.30	\$8.00	\$4.17
2,750 pounds of quartz.....	2.00	2.46	2.00	2.46	2.00	2.46
1,470 pounds of coal.....	8.00	5.30	6.00	3.95	5.00	3.30
50 pounds of electrodes.....	.04	2.00	.03	1.50	.04	2.00
0.8 kilowatt-year.....	8.30	6.62	18.66	15.00	26.66	21.30
Labor, salaries.....	^a 1	10.00	^a .80	8.00	^a 1.50	15.00
Repairs and maintenance.....		5.00		5.00		5.00
Amortization and depreciation 5 per cent each.....		2.00		2.00		2.00
Interest on \$50,000 at 6 per cent.....		1.20		1.20		1.20
General and packing.....		4.00		4.00		4.00
Total.....		45.88		50.41		60.43

^a Minimum for eight hours.

The electrode consumption is based upon the use of electrodes threaded for continuous feeding, a practice not now universally followed in ferro-alloy plants. The figures indicate that it costs \$14.55

or 31.7 per cent more to manufacture 50 per cent ferrosilicon in the United States than in Norway. This difference may be considered as due largely to the price of power. The difference in the total cost of power in Norway and the United States for a ton of 50 per cent ferrosilicon is \$14.68 per ton. Labor is also lower, but this is offset by the higher cost of raw materials abroad. The duty on ferrosilicon under the Underwood bill is 15 per cent ad valorem. The selling price of 50 per cent ferrosilicon f. o. b. Pittsburgh, Pa., is \$70 to \$75 per long ton.

USES.

The chief use for ferrosilicon is for the deoxidation of steel by the formation of silica or silicates with any free oxygen or oxides, the silica or silicates thus formed passing into the slag. A considerable amount is used for making fixed additions to steel. The grade containing 25 to 30 per cent silicon is used as a substitute for blast-furnace grades containing 10 to 12 per cent silicon. It is especially useful for direct addition to the open-hearth furnace, the cupola, or the converter, but may be added in the ladle. It is compact, in large pieces, and of sufficient specific gravity to sink readily in the bath. It reacts more energetically than does the alloy with the lower percentage of silicon and the quantities employed need not be so large. This grade is used chiefly for fixed additions of silicon.

Ferrosilicon that has 45 to 50 per cent silicon is particularly employed for deoxidation. It reacts like the 25 to 30 per cent alloy but more energetically, and is used when the greater specific gravity of the latter is not essential. It has a lower fusion point than the lower grade. It is subject to disintegration and is the only grade from which poisonous gases have been known to emanate. It is used in greater quantity than any of the other grades.

The grade containing 75 per cent silicon is employed for additions to ordinary castings. It can be used to modify a casting so as to obtain either a white or a gray casting by addition to the casting ladle. For a fixed addition in the production of silicon steels, such as transformer steel, 75 to 90 per cent ferrosilicon is employed.

Hadfield^a developed silicon steel containing 1 to 5 per cent silicon, but the usual grade contains 2.75 per cent silicon with the smallest possible amounts of carbon, manganese, and other impurities. After a double-heating treatment this steel has a higher magnetic permeability than the purest iron, but also, as is characteristic of silicon steels, has a high electrical resistance. As a result of its high permeability, it has a low hysteresis loss, and is the most desirable material known for use in electrical generating machinery.

^a Stoughton, B., *The metallurgy of iron and steel*, 1912, p. 406.

ALLOYS OF IRON AND SILICON WITH OTHER ELEMENTS.

For the use of foundries several alloys have been manufactured, such as ferrosilicomanganese-aluminum, ferrosilico-aluminum, and silicocalcium-aluminum. These alloys are intended for use as rapid deoxidizing and refining agents, and form easily fusible silicate slags of the other elements, which rise to the surface of the bath, as they are of low specific gravity. These alloys are too expensive for use in the large-scale production of ordinary steel.

Ferrosilicomanganese-aluminum is made in two grades, one containing 18 to 20 per cent silicon, 18 to 22 per cent manganese, and 9 to 12 per cent aluminum; and the other containing 9 to 11 per cent silicon, 9 to 11 per cent manganese, and 4.5 to 6 per cent aluminum, the rest in both grades being iron and carbon. The alloy is made by melting ferrosilicon-manganese and aluminum together. It is added in the ladle or molds, or in the furnace itself. It is used largely for cast steel, when the steel is cast directly into molds.

Another similar alloy is ferrosilico-aluminum, containing about 45 per cent silicon and 12 to 15 per cent aluminum, the rest being iron and carbon. It is made by alloying ferrosilicon and aluminum in the electric furnace. This alloy is intended for deoxidizing purposes. It is added in the electric steel furnace just after removal of the oxidizing slag. The alloy is in powder form and is also used in the tapping ladle for removing oxides. Sometimes it is added to the ingot molds. The electric furnace is the only furnace to which it is added for the manufacture of steel, because in the open hearth or converter there is great loss of the alloy. Ferrosilico-aluminum has not the reducing power of aluminum, but it is sufficient for the removal of most oxides.

Ferrosilicocalcium-aluminum is an electric-furnace product containing 50 to 55 per cent silicon, 18 to 22 per cent calcium, 12 to 15 per cent iron, 4 to 5 per cent aluminum, 1 to 1.25 per cent carbon, 0.35 per cent magnesium, 0.22 per cent manganese, 0.075 per cent sulphur and 0.03 per cent phosphorus. It is a strong deoxidizer and desulphurizer, and gives highly fluid slags. It is added in the electric furnace after the removal of the oxidizing slag. It can be added in the casting ladles, crucibles, or molds. Because of its high cost, its use is restricted to the manufacture of high-class steels.

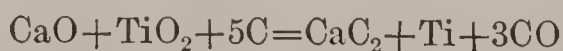
FERROTITANIUM.

Ferrotitanium has been developed for use in steel manufacture since the introduction of the electric furnace for ferro-alloy production in 1900. In 1892 Rossi obtained a patent on the reduction of titaniferous ores in the blast furnace. Later he attempted to make a ferrotitanium in the blast furnace and crucible but could not obtain a product with a high percentage of titanium. Subsequent experi-

mental work in the electric furnace resulted in the commercial establishment of the manufacture of ferrotitanium for steel making. Foreign manufacturers of ferro-alloys produced ferrotitanium on a small scale in the electric furnace. A considerable amount is made by the Goldschmidt thermit process. This grade is carbon free but contains as high as 5 per cent of aluminum, and is more expensive than the electric-furnace product.

All ferrotitanium now produced is manufactured either by reduction of ores with carbon in the electric furnace or by the Goldschmidt thermit process, most of it being made in the electric furnace. No other process gives a high enough temperature, $2,000^{\circ}\text{C}$., or over, for reduction of titanium from the oxide. In Rossi's first patent ^a on ferrotitanium, he specifies the reduction of titaniferous ores and rutile with carbon in an electric-resistance furnace to produce an alloy containing more than 5 per cent titanium and some carbon. The reduction is performed over a bath of molten iron. Another patent, ^b granted the same date, covers the use of titaniferous slag in the same manner. In 1900 he procured a patent ^c on the reduction of titanium ores by the addition of ores to a bath of molten reducing metal, such as aluminum, in the electric furnace. In 1901 Rossi ^d was granted a patent on a process for the concentration of titanium in slag by reducing the iron and silicon of a titaniferous ore with carbon at a temperature lower than the reducing point of titanium oxide.

Ferrotitanium is manufactured commercially at one plant in a Siemen's type of electric furnace of about 500 kw. capacity. A molten bath of iron from scrap iron or steel is first formed in the bottom of the furnace. Titaniferous ore, titaniferous slag, or rutile, is then mixed with carbon in the proper proportion, charged on the molten bath, and reduced in the arc. The product is tapped off and broken up for packing. If the slag is high in titanium it is run back through the furnace. The titaniferous ore contains 34.36 per cent TiO_2 , 50.53 per cent FeO , 4.14 per cent SiO_2 , and 2.20 per cent Al_2O_3 . The product obtained from this reduction without any refining contains 10 to 15 per cent titanium, 5 to 8 per cent carbon, and 0.35 to 1 per cent silicon. By refining this alloy, with rutile as a decarburizer, an alloy is made that has 10 to 15 per cent titanium, less than 1.0 per cent carbon, and 0.35 to 1 per cent silicon. In a later patent, ^e in order to reduce the volatilization loss of titanium, Rossi specifies the charging of lime on the molten iron bath, followed by a charge of titanitic acid, lime, and carbon, when reduction takes place according to the following reaction:



^a U. S. patent No. 609466, August 23, 1898.

^b U. S. patent No. 609467, August 23, 1898.

^c U. S. patent No. 648439, May 1, 1900.

^d U. S. patent No. 668266, February 19, 1901.

^e U. S. patent No. 1019528, March 5, 1912.

The titanium sinks through the lime slag to the iron, and the reducing cover of calcium carbide protects the bath from oxidation. Because of its cost rutile (TiO_2) is not so extensively used in the manufacture of ferrotitanium as a titaniferous ore. Rutile sells for \$100 to \$150 per short ton.

The presence of carbides in electric-furnace ferrotitanium have been claimed to prevent its ready solution in steel, so that it would not do the work of carbon-free ferrotitanium. However, there does not appear to be over 0.25 per cent of combined carbon in the alloy containing 9 per cent carbon. The remaining carbon is present as graphite and on addition of the ferrotitanium to the steel bath part of the carbon combines with the steel to form carbides and the rest rises to the surface and is lost in the slag.

Ferrotitanium is added to steel and cast-iron baths for the purpose of final deoxidation of the metal and its impurities, following the usual treatment for this object by means of manganese and silicon.^a Titanium has a strong affinity for oxygen and also seems to cause occluded or oxidized substances in the steel bath to separate readily. The resulting iron or steel is tougher and stronger after treatment. The alloy is added in the ladle into which the product of either the open-hearth furnace or the converter is being poured. It should be added last after carburizing and other alloys have been used, or much will be wasted. Too much slag should not be present, as titanium acts energetically with the slag. Ferrotitanium should never be preheated before being added or used in conjunction with aluminum. Little ferrotitanium is used for fixed addition to steel.

ELECTRIC SMELTING OF TUNGSTEN ORES AND THE PRODUCTION OF FERROTUNGSTEN.

HISTORY.

Ferrotungsten with a low percentage of tungsten was made by Berthier in 1834. Caron studied iron tungsten alloys in 1868 and determined that the hardness increased the percentage of tungsten. In 1866 Biermann made some ferrotungsten from tungsten trioxide and iron in the crucible. This method was used up to the introduction of the electric furnace in 1900, since which time most of the ferrotungsten has been made in the electric furnace, although small quantities, containing less than 60 per cent tungsten, are still manufactured in the crucible. The crucible product is made by the reduction of tungsten trioxide with carbon. In electric-furnace methods raw ores or concentrates are reduced.

^a Stoughton, B., *The metallurgy of iron and steel*, 1908, p. 169.

EXPERIMENTS OF STASSANO.

Stassano ^a conducted experiments on the manufacture of ferro-tungsten from wolframite concentrates in a 75-kw. electric furnace, with charcoal as a reducing agent. The concentrates, charcoal, and lime were crushed and briquetted with a 25 per cent solution of sodium silicate as a binder. Theoretically the ferrotungsten should have contained 71.5 per cent tungsten, 20.6 per cent iron, 7.1 per cent manganese, and 1 per cent silicon. The results of Stassano's experiments are tabulated below:

Constituent of ore charged:	Per cent.
WO ₃	69.8
SiO ₂	2.0
FeO.....	20.5
MnO.....	7.3
S.....	.2
P.....	Trace.
Constituent of charge:	Parts.
Wolframite.....	1,000
Charcoal.....	190
Lime.....	40
Mn ₂ SiO ₃	80

Composition of products of three experiments.

Constituent.	Proportion in experiment No. —		
	1	2	3
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
W.....	58.00	65.66	69.76
C.....	2.40	2.062	2.50
Mn.....	3.192	3.50	3.60
Si.....	1.244	1.02	1.30
P.....	Trace.	Trace.	Trace.
S.....	Trace.	Trace.	Trace.
Kilowatt-hours per pound of product.....	2.73	2.95	3.90
Kilowatt-years per ton (2,000 pounds) of product.....	.62	.67	.78

The average energy consumption was 3 kilowatt-hours per pound, or 0.68 kilowatt-year per ton. These experiments showed the possibility of reducing wolframite with carbon in the electric furnace, but did not show where the loss in tungsten occurred. It is not apparent whether the low tungsten content was the result of slag losses or the presence in the furnace of some unnoted iron which diluted the alloy.

EXPERIMENTS WITH COLORADO FERBERITE.

In 1911 the writer performed some experiments on the reduction of ferberite with carbon to produce ferrotungsten in the electric fur-

^a Stassano, E., Treatment of iron and steel in the electric furnace. Electrochem. and Met. Ind., vol. 6, 1908, p. 315.

nace.^a The furnace was one described herein in the discussion of the production of ferrochrome. Analyses of the coke and lime employed are given in the table on page 128. Pure calcium fluoride was used. The iron ore contained 94.9 per cent Fe_2O_3 , 4.10 per cent SiO_2 , 0.79 per cent CaO , 1.46 per cent Al_2O_3 , 0.05 per cent P, and 0.03 per cent S. Ferberite concentrates from Boulder County, Colo., of the following composition were obtained: 58.72 per cent WO_3 (46.6 per cent tungsten), 30.08 per cent FeO (23.4 per cent iron), 2.55 per cent MnO (0.84 per cent manganese), 4.86 per cent SiO_2 , 3.46 per cent CaO , 0.34 per cent P, and 0.20 per cent S.

In each experiment a decarburizing slag of iron ore, lime, and fluor-spar was added to the furnace when reduction of the charge had been completed and allowed to act for 10 to 20 minutes. It was tapped from the furnace with the ferrotungsten and the slag from reduction.

As shown in the table following, the alloys obtained, with two exceptions, were somewhat lower in tungsten than had been calculated. This was the result of three causes: First, the presence in the furnace of iron due to hanging of a previous run with iron ore; second, the loss of tungsten in the slag; and, third, the iron reduced from the decarburizing slag. The carbon content of all of the alloys was low for ferrotungsten, a condition clearly assisted considerably by the use of the decarburizing slag of iron oxide and lime. It was manifest that with a longer decarburization period the carbon in the alloy could have been reduced still further. The greater part of the manganese in the charge was either slagged or volatilized. The percentage of silicon was high in some of the products, but much lower when there was no iron ore in the charge. Phosphorus and sulphur were easily slagged in spite of a high percentage of both in the ore. The slags contained 4.66 to 8.64 per cent tungsten oxide (or 3.7 to 6.8 per cent tungsten), and ferrous oxide in about the same percentage.

The commercial ferrotungsten from the electric furnace contains 50 to 80 per cent tungsten and 0.5 to 4 per cent carbon.

During the experiments 32 pounds of ferrotungsten was made, or an extraction of 83.8 per cent. The electrode consumption for the two experiments in which it was determined averaged 150 pounds per ton. This figure would be considerably reduced in large-scale operations. The average energy consumption for the seven experiments was 3.46 kilowatt-hours per pound tapped, or 0.79 kilowatt-year per ton. The ferrotungsten was hard and brittle, and castings made from it were free from blowholes.

^a Keeney, R. M., The production of steels and ferro-alloys directly from ore in the electric furnace: Iron and Steel Inst., Carnegie Scholarship Memoirs, vol. 4, 1912, p. 108.

Detailed results of the experiments are tabulated below:

Results of experiments on the production of ferrotungsten directly from ore.

Experiment No.....	1	2	3	4	5	6	7
Charge:							
Ferberite.....pounds..	5.00	10.00	5.00	5.00	6.00	6.00	6.00
Hematite.....do....	1.50	3.00	1.50	1.50			
Coke.....do....	1.25	3.00	1.37	1.52	1.12	1.12	1.12
Lime.....do....	.75	1.56	.90	.95	.75	.75	.75
Fluorspar.....do....	.062	1.24	.062	.062	.062	.062	.062
Refining slag:							
Hematite.....do....	.75	.75	.75	.75	.75	.75	.75
Lime.....do....	.20	.20	.20	.20	.20	.20	.20
Fluorspar.....do....	.044	.044	.044	.044	.044	.044	.044
Composition of alloy:							
Tungsten.....per cent..	38.74	49.64	41.39	43.35	42.19	37.50	59.01
Carbon.....do....	1.77	1.82	2.05	2.51	1.66	2.04	2.02
Manganese.....do....	.32	.16	.42	.38	.16	.33	.16
Silicon.....do....	.47	.31	.24	.44	.14	.07	.09
Phosphorus.....do....	.044	.046	.042	.048	.051	.054	.020
Sulphur.....do....	.032	.068	.089	.017	.014	.021	.015
Composition of slag:							
WO ₃do....	8.64	4.14	7.20	4.24	4.00	5.28	4.88
SiO ₂do....	30.12	31.10	31.00	31.40	30.40	29.28	29.64
CaO.....do....	21.30	37.00	30.00	24.60	28.00	21.10	29.30
MgO.....do....	16.60	12.00	19.60	26.64	24.48	23.64	18.80
FeO.....do....	12.60	12.00	5.84	4.52	5.84	5.10	6.87
Al ₂ O ₃do....	12.10	5.70	5.24	6.95	6.22	11.67	7.88
Wolfram in product, calculated percentage.....	50.00	50.00	50.00	50.00	65.00	65.00	65.00
Carbon per pound W and Fe, pounds.....	.202	.222	.222	.212	.214	.214	.214
Alloy calculated, pounds.....	5.08	10.16	5.08	5.08	4.30	4.30	4.30
Alloy tapped, pounds.....	2.00	8.78	5.62	2.20	6.50	2.20	4.62
Power consumption per pound tapped, kilowatt-hours.....	4.76	2.63	2.00	5.78	1.86	5.25	1.96
Power consumption per ton tapped, kilowatt-years.....	1.09	.60	.46	1.32	.42	1.20	.45

CONCLUSIONS.

Conclusions drawn from the experiments are as follows: First, ferrotungsten can be produced directly from ferberite in the electric furnace; second, by the use of a decarburizing slag before tapping the percentage of carbon in the alloy can be kept below 2 per cent; third, manganese, silicon, phosphorus, and sulphur do not enter the ferro-alloy in high percentages; fourth, the loss of tungsten in the slag need not be excessive; and fifth, the power consumption need not exceed 3.46 kilowatt-hours per pound of ferrotungsten tapped, or 0.79 kilowatt-year per ton.

THEORY OF TUNGSTEN ALLOYS.

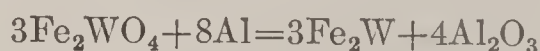
Several carbides and alloys of iron and tungsten have been isolated from ferrotungsten. Moissan ^a obtained W₂C by fusing carbon and tungsten in the electric furnace; in the same manner Williams^a found WC. Williams obtained 2Fe₃C.3W₂C by the reduction of tungsten trioxide with carbon in the electric furnace. Behrem found Fe₂W in ferrotungsten containing 50 per cent tungsten. Carnot and Goutal isolated Fe₃W. Benneville isolated Fe₄W.

^a Guillet, L., Étude industriel des alliages métallique, p. 425.

Bierman found $\text{Fe}_3\text{W}_2\text{C}$. The double carbide is not of constant composition. W_2Si_3 is known to exist, and the compound WSi is believed to exist. W_2Si_3 was obtained by Moissan synthetically in the electric furnace. The carbides W_2C and WC are both iron gray and very hard. WC fuses with difficulty.

REACTIONS.

Ferrotungsten can be made by reduction of wolframite, ferberite, or scheelite with aluminum, silicon, or carbon. Rossi^a reduced ferberite with aluminum in a Siemens type of electric furnace. An alloy containing 75.9 per cent tungsten, 21.4 per cent iron, 1.6 per cent silicon, 0.08 per cent sulphur, and 0.9 per cent carbon was made from concentrates containing 69.8 per cent WO_3 , 20.25 per cent FeO , and 5.04 per cent SiO_2 . The reaction used is as follows:

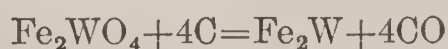


For the reduction of 100 parts of 62.3 per cent tungsten ferrotungsten from 122 parts of ferberite 24.2 parts of aluminum is necessary.

Gin has produced ferrotungsten by the reduction of scheelite with a bath of 20 per cent ferrosilicon in the electric furnace according to the following reaction:

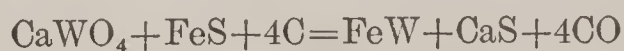


The most common method for the production of ferrotungsten is by the reduction of ferberite, wolframite, or scheelite concentrates in the electric furnace with carbon as a reducing agent, followed by a subsequent decarburization and refining. Wolframite or ferberite is readily reduced in this manner, but there is difficulty in the reduction of scheelite. Ferberite is wolframite $[(\text{FeMn})\text{WO}_4]$, in which the manganese is replaced by iron so as to give $(\text{FeFe})\text{WO}_4$. Reduction occurs with carbon according to the following reaction:



Theoretically the product contains 62.3 per cent tungsten. For the reduction of 100 parts of this alloy from 122 parts of ferberite 16.2 parts of carbon is necessary.

Scheelite is now smelted by one company with sulphide of iron and carbon in the electric furnace.^b Lime is also added to flux the silica. The following reaction takes place:



The product is said to be a ferrotungsten containing little carbon.

After much experimental work on a large scale, it seems that reduction with carbon followed by decarburization with iron ore or tung-

^a Rossi, A. J., *Ferro-alloys: Mineral Industry*, 1903, p. 693.

^b Editorial, *Electric production of ferrotungsten: Eng. and Min. Jour.*, January 20, vol. 93, 1912, p. 173.

sten concentrates is the cheapest and most efficient method of making ferrotungsten in the electric furnace. Aluminum is expensive and goes into the alloy to a certain extent, as also does silicon.

Ferrotungsten may be decarburized by covering the fused metal in the electric furnace with hammer scale, when the following reaction takes place:



The reaction is not strictly quantitative because there is always formed some ferrous tungstate.

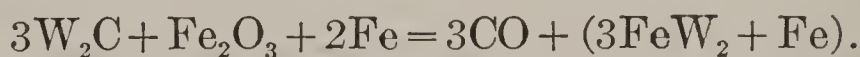
It is possible to make ferrotungsten containing 80 per cent tungsten and less than 0.5 per cent carbon by the following reaction:



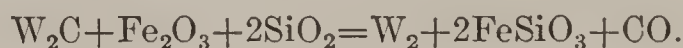
Ferberite or wolframite may be used instead of hammer scale. Slags high in tungsten are run through the reducing furnace again. By the above method carbon can not be reduced below 0.3 per cent in the alloy.

Ferrotungsten with less than 80 per cent tungsten can be made by adding metallic iron to the bath and decarburizing with iron oxide and hammer scale.

Alloys with as low a carbon content as 0.15 per cent can be made by use of the following reaction:



Decarburizing with iron oxide has a tendency to cause loss of tungsten in the slag by the formation of tungstate of iron. This loss may be reduced somewhat by making the decarburizing slag rather acid when ferrous silicate is formed, according to the following reaction:



PROCESS OF MANUFACTURE.

Ferrotungsten is manufactured from ores in three ways, as follows: First, by direct reduction with carbon in a crucible; second, by reduction in an electric furnace by some reducing agent other than carbon; and third, by direct reduction with carbon in an electric furnace.

In manufacture by the crucible process concentrates are placed in a clay-lined crucible with the proper proportions of reducing agent and flux, and heated to a high temperature in a gas-fired furnace.^b

^a Gin, G., Molybdenum and tungsten: Trans. Am. Electrochem. Soc., vol. 13, 1908, p. 48.

^b Pratt, S. R., Manufacture of metallic tungsten and ferrotungsten in the crucible: Eng. and Min. Jour., vol. 90, 1910, p. 959.

There is considerable wear on the crucible in this method. In making a 30 per cent tungsten alloy the crucible will last about three heats, but in making a 65 to 75 per cent product it lasts only one heat. Alloys with higher percentages of tungsten than this are not made in the crucible furnace.

In the electric-furnace process, ferberite, wolframite, hubuerite, or scheelite is reduced with carbon in an arc furnace. As with many expensive alloys, the process is generally an intermittent one; that is, the reduced charge is tapped from the furnace before another charge is added, or the charge is allowed to solidify and is chiseled out of the furnace. Sometimes a campaign of several days is made. The reduced alloy is either decarburized with refining slags in the same furnace or melted and refined later in another furnace. The furnaces used are similar in design to the ones described for ferrochrome production, but a more complete recovery is effected by the use of a tilting rather than a stationary furnace. Wolframite, ferberite, and scheelite are reduced easily in the electric furnace, most of the manganese being volatilized. Scheelite is more difficult to reduce and very sticky basic slags result. There is greater loss in its reduction, so that it does not command as high a price as the other ores, selling for about \$1 per unit less. The product is tapped into molds, broken on cooling, and packed into kegs or boxes.

The largest producers of tungsten ores are Australia, Colorado, California, and Argentina. The standard concentrate contains 60 per cent tungsten trioxide (WO_3). The price varies from \$5 to \$8 per unit in the 60 per cent grade; that is, if the rate was \$5, a 60 per cent concentrate would be worth \$300 per short ton. The analysis of a typical Colorado ore, ferberite, has been given previously in a discussion of the author's experiments. A typical scheelite contains 69.50 per cent WO_3 , 16.04 per cent CaO , 0.54 per cent FeO , 0.18 per cent MnO , 0.14 per cent P_2O_5 , and 12.46 per cent SiO_2 .

The results of analyses of typical ferrotungsten produced in the electric furnace are given below.

Results of analyses of typical ferrotungsten.

Analysis No.....	1	2	
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Tungsten.....	73.0	78.5	83.0
Carbon.....	3.5	1.7	.6
Iron.....	19.0	18.5	15.5
Silicon.....	.4	.3	.4
Manganese.....	3.5	.4	.2
Calcium.....	Trace.	.1	.151
Arsenic.....	.4	.1	.10
Tin.....	.1	.05	.05
Sulphur.....	.05	.04	.04
Phosphorus.....	.03	.015	.015

USES.

Tungsten was first added to steel almost entirely in the form of a powder, manufactured chemically. The powder has now been replaced by ferrotungsten to such an extent that hundreds of tons of the latter is produced annually. It is claimed by ferrotungsten manufacturers that there is less loss by oxidation and a more uniform solution of tungsten in the steel if ferrotungsten is used. The powder manufacturers claim that the alloy is not uniform in composition. There is such diversity of opinion in this respect that it is difficult to state which is the better, but the production of ferrotungsten is increasing.

Ferrotungsten is used for the addition of tungsten in the manufacture of self-hardening or high-speed steels. It is added in the furnace or in the crucible.

Most self-hardening steel contains 4 to 12 per cent of tungsten, with 2 to 4 per cent of manganese and 1.5 to 2.5 per cent of carbon. It is hard without being subjected to any heat treatment or any other process for making it so. It can not be made soft, or amended by any process known at present. All of the self-hardening steels are nonmagnetic. They are used for making tools. The cutting speed of tools made from self-hardening steel is not much, if any, greater than that of tools made from ordinary carbon tool steel, but the former take deep cuts and last longer without grinding. The 2 to 4 per cent manganese can be replaced by 1 to 2 per cent of chromium. The 12 per cent molybdenum can be replaced by 4 to 8 per cent molybdenum.

High-speed steels contain as much as 24 per cent tungsten and 0.40 to 0.80 per cent carbon. These steels do not lose their temper or toughness at a red heat, so that they are adapted for tools to be used for high-speed cutting. In the steel for cutting tools the tungsten may be replaced by one-half as much molybdenum. In this way 6 to 15 per cent molybdenum is used. A little vanadium can be added with beneficial results.

For permanent magnets a steel containing 4 to 5 per cent of tungsten and 0.5 to 0.7 per cent of carbon, if heated to a red heat and quenched in water, will retain its magnetism better than ordinary hardened carbon steel.

ELECTRIC SMELTING OF VANADIUM ORES AND THE PRODUCTION OF FERROVANADIUM.

Most of the ferrovanadium made is produced by the thermit process, by reduction of the oxide V_2O_5 in a crucible with carbon, or by reduction of vanadate of iron with carbon in a crucible. In Europe some ferrovanadium is made by the reduction of the oxide, sulphide, or vanadate of iron with carbon in the electric furnace. The largest source of vanadium is the Peru deposit of patronite, a sulphide

of vanadium containing about 60 per cent sulphur and 20 per cent vanadium. In Colorado and Utah there are large deposits of low-grade carnotite, a uranium and vanadium oxide mineral. Another ore found there is roscoelite, which contains vanadium oxide, but no uranium. The patronite ore is first roasted and then reduced by the thermit process; or in the electric furnace it may be directly reduced with lime as a desulphurizing agent and carbon as a reducing agent. The other ores are treated by chemical or ore-dressing processes to obtain the vanadium as oxide or vanadate of iron, and are then reduced with carbon in a combustion or electric furnace or by the thermit method. With oxides the electric furnace method is similar to the production of ferrotungsten with carbon as a reducing agent. The sulphide reduction is similar to that of molybdenite with lime and carbon. In several patented methods silicon is used as a reducing agent in the electric furnace. The process as performed in the electric furnace has no novel features beyond those stated regarding chromium, tungsten, and molybdenum. Decarburization is with oxide of iron ore or oxide ore of vanadium. Vanadium is rather difficult to reduce, and as a result excess carbon is used which is absorbed up to about 6 per cent by the ferrovanadium. The electric furnace product is made in 50 per cent and 25 per cent grades, with carbon varying from 1 to 4 per cent. Below are the results of analyses of these grades:

Results of analyses of ferrovanadium.

Constituent.	Grade with 50 per cent vanadium.	Grade with 25 to 30 per cent vanadium.
Vanadium.....	55.00	34.10
Iron.....	40.00	64.10
Carbon.....	4.00	1.42
Silicon.....	.30	.12
Aluminum.....	.10	.12
Manganese.....	.30	.12
Sulphur.....	.03	.03
Phosphorus.....	.04	.009

Vanadium acts like titanium as a cleanser and deoxidizer of a steel bath, and is also used to add as much as about 0.35 per cent vanadium to the steel itself. It is used after ferrosilicon and ferromanganese have been added, so as to prevent loss, and causes deoxidation beyond the point obtained by these alloys. It melts without much difficulty and attacks the oxides present. Incorporated in the steel in proportions up to 0.30 per cent it toughens and adds a great tensile strength to the steel.

GLOSSARY.

Inasmuch as this bulletin has been prepared for the purpose of giving information to those who are not familiar with electrical work, the following glossary, taken mainly from Kent's "Mechanical Engineer's Pocket-Book" (eighth edition) has been prepared:

UNITS USED IN ELECTRICAL CALCULATIONS.

Ampere.—The unit of current strength, or rate of flow, represented by I .

Volt.—The unit of electromotive force, electrical pressure, or difference of potential, represented by E .

Ohm.—The unit of resistance, represented by R .

Coulomb (or ampere-second).—The unit of quantity, Q .

Ampere-hour.—3,600 coulombs, Q' .

Joule (volt-coulomb).—The unit of energy or work, W .

Watt (ampere-volt, or volt-ampere).—The unit of power, P .

Farad.—The unit of capacity, represented by C .

Henry.—The unit of inductance, represented by L .

If letters are used to represent the units, the relations between them may be expressed by the following formulas, in which t represents 1 second and T 1 hour:

$$I = \frac{E}{R}, \quad Q = It, \quad Q' = IT, \quad C = \frac{Q}{E}, \quad W = QE, \quad P = IE.$$

As these relations contain no coefficient other than unity, the letters may represent any quantities given in terms of those units. For example, if E represents the number of volts of electromotive force, and R the number of ohms of resistance in a circuit, then their ratio, $E \div R$, will give the number of amperes of current strength in that circuit.

The above six formulas can be combined by substitution or elimination, so as to give the relations between any of the quantities. The most important of these are the following:

$$Q = \frac{E}{R}t, \quad C = \frac{I}{E}t, \quad W = IEt = \frac{E^2}{R}t = I^2Rt = Pt$$

$$E = IR, \quad R = \frac{E}{I}, \quad P = \frac{E^2}{R} = I^2R = \frac{W}{t} = \frac{QE}{t}.$$

RELATIONS OF VARIOUS UNITS.

1 ampere.....	=1 coulomb per second.
1 volt-ampere.....	=1 watt=1 volt-coulomb per second.
1 watt.....	= $\begin{cases} 0.7373 \text{ foot-pound per second.} \\ 0.0009477 \text{ heat unit per second (F).} \\ 1/746 \text{ of 1 horsepower.} \end{cases}$
1 joule.....	= $\begin{cases} 0.7373 \text{ foot-pound.} \\ \text{Work done by 1 watt in 1 second.} \\ 0.0009477 \text{ heat unit.} \end{cases}$
1 British thermal unit.....	=1055.2 joules.
1 kilowatt, or 1,000 watts.....	=1,000/746, or 1.3405 horsepower.
1 kilowatt-hour.....	=1.3405 horsepower-hours.
1,000 volt-ampere hours.....	=2,645,200 foot-pounds.
1 British Board of Trade unit.....	=3412 heat units.
1 horsepower.....	= $\begin{cases} 746 \text{ watts}=746 \text{ volt-amperes.} \\ 33,000 \text{ foot-pounds per minute.} \end{cases}$

The ohm, ampere, and volt are defined in terms of one another as follows: Ohm, the resistance of a conductor through which a current of 1 ampere will pass when the electromotive force is 1 volt. Ampere, the quantity of current that will flow through a resistance of 1 ohm when the electromotive force is 1 volt. Volt, the electromotive force required to cause a current of 1 ampere to flow through a resistance of 1 ohm.

Equivalent values of electrical and mechanical units.

Unit.	Equivalent value in other units.	Unit.	Equivalent value in other units.
1 kilowatt-hour.	1,000 watt hours. 1.34 horsepower-hours. 2,654,200 foot-pounds. 3,600,000 joules. 3,412 heat units. 367,000 kilogram-meters. 0.235 pound of carbon oxidized with perfect efficiency. 3.53 pounds of water evaporated from and at 212° F. 22.75 pounds of water raised from 62° to 212° F.	1 foot-pound...	1.356 joules. 0.1383 kilogram-meter. 0.00000377 kilowatt-hour. 0.001285 heat units. 0.000005 horsepower-hour.
1 horsepower-hour.	0.746 kilowatt-hour. 1,980,000 foot-pounds. 2,545 heat units. 273,740 kilogram-meters. 0.175 pound of carbon oxidized with perfect efficiency. 2.64 pounds of water evaporated from and at 212° F. 17 pounds of water raised from 62° to 212° F.	1 watt.....	1 joule per second. 0.00134 horsepower. 3.412 heat units per hour. 0.7373 foot-pound per second. 0.0035 pound of water evaporated per hour. 44.24 foot-pounds per minute. 8.19 heat units per square foot per minute. 6,371 foot-pounds per square foot per minute. 0.193 horsepower per square foot. 1,055 watt-seconds. 778.6 kilogram-meters. 0.000293 kilowatt-hour. 0.000393 horsepower-hour. 0.0000688 pound of carbon oxidized. 0.001036 pound of water evaporated from and at 212° F.
1 kilowatt.....	1,000 watts. 1.34 horsepower. 2,654,200 foot-pounds per hour. 44,240 foot-pounds per minute. 737.3 foot-pounds per second. 3,412 heat units per hour. 56.9 heat units per minute. 0.948 heat unit per second. 0.2275 pound of carbon oxidized per hour. 3.53 pounds of water evaporated per hour from and at 212° F.	1 heat unit....	0.122 watt per square inch. 0.0176 kilowatt per square foot. 0.0236 horsepower per square foot. 7.233 foot-pounds. 0.0000365 horsepower-hour. 0.00000272 kilowatt-hour. 0.0093 heat unit. 14,544 heat units. 1.11 pounds of anthracite coal oxidized. 2.5 pounds of dry wood oxidized. 21 cubic feet of illuminating gas. 4.26 kilowatt-hours. 5.71 horsepower-hours. 11,315,000 foot-pounds. 15 pounds of water evaporated from and at 212° F.
1 horsepower...	746 watts. 0.746 kilowatt. 33.000 foot-pounds per minute. 550 foot-pounds per second. 2,545 heat units per hour. 42.4 heat units per minute. 0.707 heat unit per second. 0.175 pound of carbon oxidized per hour. 2.64 pounds of water evaporated per hour from and at 212° F.	1 kilogram-meter.	0.283 kilowatt-hour. 0.379 horsepower-hour. 965.7 heat units. 103,900 kilogram-meters. 1,019,000 joules. 751,300 foot-pounds. 0.0664 pound of carbon oxidized.
1 joule.....	1 watt second. 0.000000278 kilowatt-hour. 0.102 kilogram-meter. 0.0009477 heat unit. 0.7373 foot-pound.	1 pound of carbon oxidized with perfect efficiency.	
		1 pound of water evaporated from and at 212° F.	

LAWS OF ELECTRICAL RESISTANCE.

The resistance, *R*, of any conductor varies directly as its length, *l*, and inversely as its section area, *s*, or $R \propto l \div s$.

If *r* = the resistance of a conductor 1 unit in length and 1 square unit in sectional area, $R = rl \div s$. The common unit of length for electrical calculations in English measure is the foot, and the unit of area of wires is the circular mil, which equals the area of a circle 0.001 inch in diameter. One mil-foot is equivalent to an object 1 foot long and 1 circular mil in cross section. Resistance of 1 mil-foot of soft copper wire at 51° F.=10 international ohms.

Example.—What is the resistance of a wire 1,000 feet long and 0.1 inch in diameter?
0.1 inch in diameter=10,000 circular mils.

$R = rl \div s = 10 \times 1,000 \div 10,000 = 1 \text{ ohm.}$

Specific resistance, also called resistivity, is the resistance of a material of unit length and section as compared with the resistance of soft copper.

Conductivity is the reciprocal of specific resistance, or the relative conducting power compared with copper taken at 100.

Electrical conductivity of different metals and alloys.

Pure silver.....	100	Swedish iron.....	16
Pure copper.....	100	Pure Banca tin.....	15.45
Telegraphic siliceous bronze.....	98	Aluminum bronze (10 per cent).....	12.6
Alloy of one-half copper and one-half silver..	86.65	Siemens steel.....	12
Pure gold.....	78	Pure platinum.....	10.6
Silicide of copper, 4 per cent Si.....	75	Copper with 10 per cent nickel.....	10.6
Telephonic siliceous bronze.....	35	Pure lead.....	8.88
Pure zinc.....	29.9	Bronze with 20 per cent tin.....	8.4
Brass with 35 per cent zinc.....	21.5	Pure nickel.....	7.89
Phosphor tin.....	17.7	Phosphor bronze, 10 per cent tin.....	6.5
Alloy of one-half gold and one-half silver....	16.12	Antimony.....	3.88

Conductivity of aluminum.—J. W. Richards (Jour. Franklin Inst., Mar., 1897) gives for hard-drawn aluminum of purity 98.5, 99, 99.5, and 99.75 per cent, respectively, a conductivity of 55, 59, 61, and 63 to 64 per cent, copper being 100 per cent. The Pittsburgh Reduction Co. claims that its purest aluminum has a conductivity of over 64.5 per cent. (Eng. News, Dec. 17, 1896.)

German silver.—The resistance of German silver depends on its composition. Matthiessen gives it as nearly 13 times that of copper, with a temperature coefficient of 0.0004433 per degree centigrade. Weston, however (Proc. Electrical Cong., 1893, p. 179), has found copper-nickel-zinc alloys (German silver) that had a resistance of nearly 28 times that of copper and a temperature coefficient of about one-half that given by Matthiessen.

Conductors and insulators in order of their value.

CONDUCTORS.

All metals.	Metallic ores.
Well-burned charcoal.	Animal fluids.
Graphite.	Living vegetable substances.
Acid solutions.	Moist earth.
Saline solutions.	Water.

INSULATORS (NONCONDUCTORS).

Dry air.	Ebonite.	Jet.
Shellac.	Gutta percha.	Glass.
Paraffin.	India rubber.	Mica.
Amber.	Silk.	Porcelain.
Resins.	Dry paper.	Oils.
Sulphur.	Parchment.	
Wax.	Dry leather.	

According to Culley, the resistance of distilled water is 6,754 million times as great as that of copper. Impurities in water decrease its resistance.

RESISTANCE VARIES WITH TEMPERATURE.

For every degree centigrade, the resistance of copper increases about 0.4 per cent, or for every degree Fahrenheit, 0.2222 per cent. Thus, a piece of copper wire having a resistance of 10 ohms at 32° would have a resistance of 11.11 ohms at 82° F.

The following table shows the amount of resistance of a few substances used for various electrical purposes by which 1 ohm is increased by a rise of temperature of 1° C.

Platinoid.....	0.00021	Gold, silver.....	0.00065
Platinum silver.....	.00031	Cast iron.....	.00080
German silver.....	.00044	Copper.....	.00400

OHM'S LAW.

This law expresses the relation between the three fundamental units of resistance, electrical pressure, and current. It is:

$$\text{Current} = \frac{\text{electrical pressure}}{\text{resistance}}; I = \frac{E}{R}; \text{whence } E = IR \text{ and } R = \frac{E}{I}.$$

In terms of the units of the three quantities,

$$\text{Amperes} = \frac{\text{volts}}{\text{ohms}}; \text{volts} = \text{amperes} \times \text{ohms}; \text{ohms} = \frac{\text{volts}}{\text{amperes}}.$$

EXAMPLES.

1. If the source has an effective electrical pressure of 100 volts, and the resistance is 2 ohms, what is the current?

$$I = \frac{E}{R} = \frac{100}{2} = 50 \text{ amperes}$$

2. What pressure will give a current of 50 amperes through a resistance of 2 ohms?

$$E = IR = 50 \times 2 = 100 \text{ volts.}$$

3. What resistance is required to obtain a current of 50 amperes when the pressure is 100 volts?

$$R = E \div I = 100 \div 50 = 2 \text{ ohms.}$$

Ohm's law applies equally to a complete electrical circuit and to any part thereof.

HEAT GENERATED BY A CURRENT.

Joule's law shows that the heat developed in a conductor is directly proportional, first, to its resistance; second, to the square of the current strength; and, third, to the time during which the current flows, or $H = I^2 R t$. Since $I = E \div R$,

$$I^2 R t = \frac{E}{R} I R t = E I t = E \frac{E t}{R} = \frac{E^2 t}{R}$$

Or, heat = current² × resistance × time

= electromotive force × current × time

= electromotive force² × time ÷ resistance

Q = quantity of electricity flowing = It , or $(Et \div R)$

$H = EQ$, or heat = electromotive force × quantity.

The electromotive force here is that causing the flow, or the difference in potential between the ends of the conductor.

The electrical unit of heat or "joule" equals 10^7 ergs, equals heat generated in 1 second by a current of 1 ampere flowing through a resistance of 1 ohm, equals 0.239 gram of water raised 1° C. $H = I^2 R t \times 0.239$ gram calories = $I^2 R t \times 0.0009478$ British thermal unit.

In electric lighting the energy of the current is converted into heat in the lamps. The resistance of the lamp is made great so that the required quantity of heat may be developed, while in the wire leading to and from the lamp the resistance is made as small as is commercially practicable, so that as little energy as possible may be wasted in heating the wire.

SELECTED BIBLIOGRAPHY.

BOOKS.

- ASHCROFT, E. A. Study of electrothermal and electrolytic industries. New York, 1909. Intended to supply information about all industrially successful electrochemical and electrometallurgical industries.
- BORCHERS, W. Electric furnaces; the production of heat from electrical energy, and the construction of electric furnaces. Translated by H. G. Solomon, London, 1908.
- GUILLET, S. Étude industriel des alliages métallique. Paris.
- HAANEL, E. Report of commission appointed to investigate different electrothermic processes for smelting iron ores and making steel in Europe. Canada, Department of the Interior, Mines Branch, 1904.
- Report on electric shaft furnace at Domnarfvet, Sweden. Canada, Department of the Interior, Mines Branch, 1909.
- Recent advances in electric smelting. Bulletin No. 3, Canada, Department of the Interior, Mines Branch, 1910.
- Report on experiments at Sault Sainte Marie, Ontario, under government auspices on smelting of Canadian iron ores by the electrothermic process. Canada, Department of the Interior, Mines Branch, 1907.
- HAVARD, F. T. Refractories and furnaces. New York, 1912.
- INGALLS, W. R. The metallurgy of zinc.
- KERSHAW, J. B. C. Electrometallurgy. London, 1908.
- MINET, A. The production of aluminium and its industrial use. Translated with addition by L. Waldo, New York, 1905.
- MOISSAN, H. The electric furnace. Translated by V. Lenher, 1904.
- NEUMANN, B. Elektrometallurgie des Eisens, W. Knapp, Halle, Germany, 1907.
- ROBIN, F. Traite de Metallographie. Paris, 1912.
- RODENHAUSER, W., and SCHOENAWA, I. Electric furnaces in the iron and steel industry. Translated by C. H. Vom Baur. 1913.
- STANSFIELD, A. The electric furnace, its evolution, theory, and practice. New York, 1907.
- THOMPSON, M. DE KAY. Applied electrochemistry. New York, 1911.
- WRIGHT, J. Electric furnaces and their industrial applications. New York, 1904.

PUBLISHED PAPERS AND ARTICLES.

ALUMINUM.

- BURGESS, C. F., and HAMBUECHEN, C. Some laboratory observations on aluminum. Electrochem. Ind., vol. 1, 1902, p. 165.
- ENGINEERING AND MINING JOURNAL. Electric reduction of aluminum. Vol. 93, 1912, p. 641.
- HAMBUECHEN, C. See Burgess, C. F.
- HUTTON, R. S., and PETAVEL, J. E. On direct reduction of alumina with carbon. Electrochem. and Met. Ind., vol. 6, 1908, p. 104.
- MINERAL INDUSTRY. Aluminum. Vol. 19, 1910, p. 18, vol. 20, 1911, p. 22.
- NEUMANN, B., and OLLSEN, H. Production of aluminium as a laboratory experiment. Met. and Chem. Eng., vol. 7, 1910, p. 185.
- OLLSEN, H. See Neumann, B.
- PETAVEL, J. E. See Hutton, R. S.

- RICHARDS, J. W. Present metallurgy of aluminium. *Electrochem. Ind.*, vol. 1, 1902, p. 158.
- Aluminium nitride. *Trans. Am. Electrochem Soc.*, vol. 23, 1913, p. 351.
- RICHARDSON, H. Some observations on laboratory production of aluminium. *Trans. Am. Electrochem. Soc.*, vol. 19, 1911, p. 159.
- THOMPSON, M. DE KAY. The electrolytic reduction of aluminium as a laboratory experiment. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 14.
- TUCKER, S. A. The preparation of aluminium in the laboratory. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 315.

ARC FURNACE.

- ALEXANDER, W. A., TUCKER, S. A., and HUDSON, H. K. Relative efficiency of the arc and resistance furnace for the manufacture of calcium carbide. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 411.
- HANSEN, C. A. Small experimental Héroult furnace. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 206.
- HUDSON, H. K. See Alexander, W. A.
- THOMPSON, M. DE KAY. Preparation and properties of calcium carbide. *Trans. Am. Electrochem. Soc.*, vol. 16, 1909, p. 197.
- TONE, F. T. Production of silicon in the electric furnace. *Trans. Am. Electrochem. Soc.*, vol. 7, 1905, p. 243.
- TUCKER, S. A. The preparation of silicon in the laboratory. *Met. and Chem. Eng.*, vol. 8, 1910, p. 19.
- See also Alexander, W. A.
- WEEDON, W. S. A contribution to the study of the electric arc. *Trans. Am. Electrochem. Soc.*, vol. 5, 1904, p. 171.
- The titanium arc. *Trans. Am. Electrochem. Soc.*, vol. 16, 1909, p. 217.
- WHITNEY, W. R. Arcs. *Trans. Am. Electrochem. Soc.*, vol. 7, 1905, p. 291.

CALCIUM AND MAGNESIUM.

- BADGER, W. L., and FRARY, F. C. The preparation of calcium. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 185.
- BICKNELL, H. R., FRARY, F. C., and TRONSON, C. A. Efficiency in the electrolytic production of metallic calcium. *Trans. Am. Electrochem. Soc.*, vol. 18, 1910, p. 117.
- FRARY, F. C. See Badger, W. L.; also Bicknell, H. R.
- JOHNSON, A. R. The electrolytic preparation of calcium. *Trans. Am. Electrochem. Soc.*, vol. 18, 1910, p. 125.
- JOUARD, F. L., and TUCKER, S. A. The electrolytic preparation of magnesium. *Trans. Am. Electrochem. Soc.*, vol. 17, 1910, p. 249.
- TRONSON, C. A. See Bicknell, H. R.
- TUCKER, S. A. See Jouard, F. L.

COPPER.

- HAANEL, E. Report of the commission appointed to investigate the different electrothermic processes for the smelting of iron ores and the making of steel in operation in Europe. Canada, Department of Interior, Mines Branch, 1904. Report on electric copper smelting by Vatier.
- KEENEY, R. M., and LYON, D. A. The smelting of copper ores in the electric furnace. *Am. Inst. Min. Eng.*, No. 80, Aug., 1913, p. 2151; *Met. and Chem. Eng.*, vol. 11, 1913, p. 522.
- LYON, D. A. See Keeney, R. M.
- RICHARDS, J. W. Metallurgical calculations. Electric smelting of copper ores. *Electrochem. and Met. Ind.*, vol. 5, 1907, p. 496.
- The electric furnace in nonferrous metallurgy. *Met. and Chem. Eng.*, vol. 8, 1910, p. 233.

- SCHILOWSKI, J. La fusion électrique de cuivre et des produits intermédiaires de fonderies de cuivre. *Rev. de Mét.*, vol. 9, 1912, p. 205.
- STEPHAN, M. Einiges über die Erzeugung von Metallen im elektrischen Ofen. *Metall. und Erz.*, vol. 1, 1912, p. 11.
- WEEKS, C. A. Melting nonferrous metals in an electric furnace. *Met. and Chem. Eng.*, vol. 9, 1911, p. 363.
- WOLKOFF, W. Electric smelting of copper sulphide ores. *Metallurgie*, vol. 7, 1910, p. 99.

ELECTRODES.

- CLOCHER, W. Manufacture of carbons. *Met. and Chem. Eng.*, vol. 9, 1911, p. 137.
- COLLENS, C. L. Graphite electrodes in electrometallurgical processes. *Trans. Am. Electrochem. Soc.*, vol. 1, 1902, p. 53.
- FITZGERALD, F. A. J. On testing of carbon electrodes. *Trans. Am. Electrochem. Soc.*, vol. 2, 1902, p. 43.
- On carbons for electrometallurgy. *Trans. Am. Electrochem. Soc.*, vol. 11, 1907, p. 317.
- FITZGERALD, F. A. J., and HINCKLEY, A. T. Experiments with furnace electrodes. *Trans. Am. Electrochem. Soc.*, vol. 23, 1913, p. 333.
- FORSELL, J. Current densities and energy losses in electrodes. *Met. and Chem. Eng.*, vol. 8, 1910, p. 26.
- HANSEN, C. A. Furnace electrode losses. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 279; vol. 16, 1909, p. 329.
- Furnace electrode losses. *Electrochem. and Met. Ind.*, vol. 7, 1909, pp. 358, 389.
- HERING, C. Laws of electrode losses in electric furnaces. *Trans. Am. Electrochem. Soc.*, vol. 16, 1909, p. 265.
- A new method of measuring mean thermal and electrical conductivities of furnace electrodes. *Trans. Am. Electrochem. Soc.*, vol. 16, 1909, p. 317.
- Determination of the constants of materials for furnace electrodes. *Trans. Am. Electrochem. Soc.*, vol. 17, 1910, p. 151.
- Empirical laws of furnace electrodes. *Trans. Am. Electrochem. Soc.*, vol. 17, 1910, p. 171.
- Method for determining thermal conductivities. *Trans. Am. Electrochem. Soc.*, vol. 18, 1910, p. 213.
- Furnace electrode losses. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 400.
- Properties and behavior of furnace electrodes. *Met. and Chem. Eng.*, vol. 7, 1909, p. 442; Discussion, vol. 9, 1910, pp. 42, 67, 665; vol. 10, 1910, pp. 128, 154, 206.
- Electrode efficiency of furnaces. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 473.
- Chilling and heating of electrodes. *Met. and Chem. Eng.*, vol. 8, 1910, p. 188.
- Electrode design. *Proc. Am. Inst. Elec. Eng.*, vol. 29, 1910, p. 285.
- HERING, C., and KENNELLY, A. E. Furnace electrode losses. *Met. and Chem. Eng.*, vol. 8, 1910, p. 238.
- HINCKLEY, A. T. See Fitzgerald, F. A. J.
- KENNELLY, A. E. The modification in Hering's law of electrodes by including variations in electric and thermal resistivity. *Proc. Am. Inst. Elec. Eng.*, vol. 29, 1910, p. 465.
- Electrode-holder construction for electric furnaces. *Met. and Chem. Eng.*, vol. 11, 1913, p. 321.
- Manufacture of carbon electrodes. *Met. and Chem. Eng.*, vol. 9, 1911, pp. 42, 67, 111, 137, 221, 377, 559, 665.
- Production of electrodes. *Met. and Chem. Eng.*, vol. 8, 1910, p. 291.
- See also Hering, C.

- MOISSAN, H. Electrodes. *Compt. Rend.*, vol. 135, p. 1902.
- ROEBER, E. F. Electrode losses in electric furnaces. *Trans. Am. Electrochem. Soc.*, vol. 16, 1909, p. 363.
- ROUSH, G. A. Manufacture of carbons for steel furnaces. *Jour. Ind. and Eng. Chem.*, vol. 1, 1909, p. 286.
- TURNBULL, R. Furnace electrodes practically considered. *Trans. Am. Electrochem. Soc.*, vol. 21, 1912, p. 397.

ELECTROMECHANICAL FORCES.

- HARDEN, J. The "pinch" effect in electric furnaces of the induction type. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 478.
- HERING, C. Formulas for the pinch phenomenon. *Met. and Chem. Eng.*, vol. 9, 1911, p. 86.
- A practical limitation of resistance furnaces; the pinch phenomenon. *Trans. Am. Electrochem. Soc.*, vol. 21, 1907, p. 329.
- The working limit in electric furnaces due to the "pinch" phenomenon. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 255.
- A new type of electric furnace. *Trans. Am. Electrochem. Soc.*, vol. 19, 1911, p. 255.
- Electric furnaces for molten materials. *Met. and Chem. Eng.*, vol. 9, 1911, p. 37; *Trans. Am. Electrochem. Soc.*, vol. 19, 1911, p. 255.
- An imperfection in the usual statement of the fundamental law of electromagnetic induction. *Proc. Am. Inst. Elec. Eng.*, vol. 27, 1908, p. 134.
- NORTHRUP, E. F. Some newly observed manifestations of forces in the interior of an electric conductor. *Physical Rev.*, vol. 24, No. 6, 1907, p. 474.
- SCOTT, E. K. The Hering "pinch-effect" furnace. *Trans. Faraday Soc.*, vol. 7, 1912, p. 202.
- UNGER, M. The action of electromechanical forces in the bath of an induction furnace. *Met. and Chem. Eng.*, vol. 10, 1912, p. 263.

ELECTROLYSIS.

- ASHCROFT, E. A. Factory scale experiments with fused electrolytes. *Electrochem. and Met. Ind.*, vol. 4, 1906, pp. 143, 178, and 357.
- BEEKMAN, J. W. An electrolytic furnace method for producing metals. *Trans. Am. Electrochem. Soc.*, vol. 19, 1911, p. 171.
- BURGESS, C. F. Electrolytic refining as a step in the production of steel. *Trans. Am. Electrochem. Soc.*, vol. 19, 1911, p. 181.
- CHANCE, E. M. A new method for the electrolytic winning and refining of metals. *Trans. Am. Electrochem. Soc.*, vol. 17, 1910, p. 235.
- GOODWIN, H. M., and MAILEY, R. D. On the density, electrical conductivity, and viscosity of fused salts. *Trans. Am. Electrochem. Soc.*, vol. 11, 1907, p. 211.
- MAILEY, R. D. See Goodwin, H. M.

FERRO-ALLOYS.

- BALLAGH, J. C., and IWAI, K. Investigation of ferroboron. *Min. and Sci. Press*, vol. 99, 1909, p. 185.
- BENNETT, S. R., COPEMAN, S. M., and HAHE, H. W. Manufacture and transport of ferrosilicon. *Yellow book of British Local Government Board*; *Met. and Chem. Eng.*, vol. 8, 1910, p. 133; *Iron and Coal Trades Rev.*, vol. 80, 1910, p. 45.
- CARNOT, and GOUTAL. Recherches sur l'état où se trouvent le silicium et le chrome dans les produits sidérurgiques. *Compt. Rend.*, vol. 126, 1898, p. 1240.
- CATANI, R. The application of electricity in the metallurgy of Italy. *Jour. Iron and Steel Inst.*, vol. 84, No. 2, 1911, p. 215; *Met. and Chem. Eng.*, vol. 9, 1911, p. 642.
- CHAPLET, M. Les alliages ferrométalliques, *Rev. de Met.*, vol. 6, 1909, p. 739.

- CONRAD, W. Electric furnaces for the manufacture of calcium carbide and ferro-silicon. *Electrochem. and Met. Ind.*, vol. 6, 1908, p. 397.
- CONRAD, W., and PICK, W. Die Herstellung von Hochprozentigem Ferrosilizium im elektrischen Ofen. Pub. by W. Knappe, Halle, Germany, 1909; *Rev. de Met.*, vol. 9, 1912, p. 362.
- COPEMAN, S. M. See Bennett, S. R.
- ELECTROCHEMICAL INDUSTRY. Ferrosilicon made in the electric furnace. Vol. 2, 1904, p. 122.
- ELECTROCHEMICAL AND METALLURGICAL INDUSTRY. Ferro-alloys. Vol. 4, 1906, p. 247.
- ENGINEER (London). The manufacture and use of ferro-alloys. Vol. 105, 1908, pp. 80 and 105.
- ESCARD, M. Ferrosilicium. *La Lumière Electrique*. Mar. 6, 1909.
- GENIE CIVIL. La fabrication du ferromanganese en France. Feb. 10, 1906.
- GIN, G. Decarburization of ferro-alloys. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 225.
- Ferrosilicium. *L'Eclairage Electrique*, vol. 27, May 4, 1900.
- Memoirs on the methods of treatment of simple and complex ores of molybdenum, tungsten, uranium, and vanadium. *Trans. Am. Electrochem. Soc.*, vol. 12, 1907, p. 411; vol. 13, 1908, p. 481; vol. 16, 1909, p. 393.
- Note upon the manufacture of silicovanadium. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 229.
- Preparation of ferrovanadium by electrolysis. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 227.
- GIROD, P. Les alliages ferrométalliques. Société des Ingeneurs Civils de France, Nov., 1906.
- GOUTAL, see Carnot.
- GREENWOOD, H. C., SLADE, R. E., and PRING, J. N. Reduction of refractory oxides, production of ferro-alloys, and formation of carbides. *Trans. Chem. Soc.*, vol. 93, 1908, p. 1484, *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 119.
- GUICHARD, M. Production of molybdenum from molybdenite. *Compt. Rend.*, vol. 122, p. 1270.
- HADFIELD, R. A. Alloys of iron and tungsten. *Jour. Iron and Steel Inst.*, vol. 64, No. 2, 1903, p. 14.
- HAHE, H. W., see Bennett, S. R.
- HAUGHTON, E. Ferro-alloys in the foundry. *Electrochem. and Met. Ind.*, vol. 5, 1907, p. 9.
- HUTTON, R. S. Recent advances in the electrometallurgy of iron and steel. *Soc. Chem. Ind.*, vol. 24, pt. 1, 1905, p. 590.
- IRON TRADE REVIEW. Ferrovanadium. Vol. 51, 1912, p. 275.
- IWAI, K. See Ballagh, J. C.
- JAKOKI, J. Ferromanganese in Hochofen. *Stahl und Eisen.*, vol. 29, 1909, p. 1191.
- KEENEY, R. M. The production of steels and ferro-alloys directly from ore in the electric furnace. *Jour. Iron and Steel Inst.*, vol. 4, 1912, p. 108.
- Electric smelting of chromium, tungsten, molybdenum, and vanadium ores. *Trans. Am. Electrochem. Soc.*, vol. 24, 1913, p. 167; *Met. and Chem. Eng.*, vol. 11, 1913, p. 585.
- KEENEY, R. M., and LEE, G. M. The direct production of steels and ferro-alloys from ore in the electric furnace. *West. Chem. and Metal*, vol. 6, 1910, pp. 269, 323, 347.
- KEENEY, R. M., and LYON, D. A. Possible application of the electric furnace to western metallurgy. *Trans. Am. Electrochem. Soc.*, vol. 24, 1913, p. 119; *Met. and Chem. Eng.*, vol. 11, 1913, p. 577.

- KELLER, C. A. The electric furnace and the manufacture of ferro-alloys. *Jour. Iron and Steel Inst.*, vol. 63, 1903, No. 1, p. 162.
- LEBEAU, P. Sur les gas toxiques dégagee par les ferrosilicium sous l'action l'air humide. *Rev. de Métal.*, vol. 6, 1909, p. 907.
- LEE, G. M. See Keeney, R. M.
- LEHNER, W. Herstellung von Molybdenum. *Metallurgie*, vol. 3, 1906, p. 549.
- LOUIS, J. La fabrication des ferrosilicium au four électrique. *Jour. du Four Electrique et de Lumière*, Oct. 1, 1910, p. 415.
- LYON, D. A. See Keeney, R. M.
- METALLURGICAL AND CHEMICAL ENGINEERING. Electric smelting of nickel ore. Vol. 8, 1910, p. 277.
- Swedish methods of packing ferrosilicon. Vol. 11, 1913, p. 87.
- MOISSAN, H. *Compt. Rend.*, vol. 121, p. 621.
- MORRISON, W. L. Electric-furnace treatment of nickel ore and the development of a commercial process. *Trans. Am. Electrochem. Soc.*, vol. 20, 1911, p. 315; *Met. and Chem. Eng.*, vol. 9, 1911, p. 546.
- NEUMANN, B. Herstellung von Ferro-liegierung im elektrischen Ofen. *Stahl und Eisen*, vol. 28, 1908, p. 356.
- PICK, W. See Conrad, W.
- PETAVEL. Ferrosilicium. *Journal d'Electrolyse*, vol. 3, June 3, 1904.
- PRATT, L. R. Manufacture of metallic tungsten and ferrotungsten in the crucible. *Electrochem. and Met. Jour.*, vol. 90, 1910, p. 959.
- PRING, J. N. See Greenwood, H. C.
- ROSSI, A. J. Ferro-alloys. *Min. Ind.*, vol. 12, 1903, p. 693.
- Exhibit of ferrometals made electrically, and of other electric-furnace products. *Trans. Am. Electrochem. Soc.*, vol. 5, 1904, p. 275.
- On the manufacture of ferro-alloys in general and of ferrotitanium in particular in the electric furnace. *Electrochem. Ind.*, vol. 1, 1903, p. 523.
- SCHOLL, G. P. Manufacture of ferro-alloys in the electric furnace. *Electrochem. Ind.*, vol. 2, 1904, pp. 349, 395, 449.
- SCROEDER, F. Remelting of ferromanganese in the electric furnace and the use of molten ferromanganese for deoxidization. *Met. and Chem. Eng.*, vol. 9, 1910, p. 640.
- SJÖSTEDT, E. A. Electric-smelting experiments for the manufacture of ferronickel from pyrrhotite. *Trans. Am. Electrochem. Soc.*, vol. 5, 1904, p. 233.
- SLADE, R. E. See Greenwood, H. C.
- SLOCUM, C. V. Titanium in iron and steel. *Trans. Am. Electrochem. Soc.*, vol. 20, 1911, p. 265.
- STANSFIELD, A. The electrothermic production of steel from iron ore. *Trans. Can. Min. Inst.*, vol. 10, 1907, p. 128.
- Tool steel direct from ore in an electric furnace. *Trans. Can. Min. Inst.*, vol. 13, 1910, p. 151.
- STASSANO, E. Treatment of iron and steel in the electric furnace. *Electrochem. and Met. Ind.*, vol. 6, 1908, p. 35.
- STEINHART, D. J. Notes on metals and their ferro-alloys used in the manufacture of alloy steels. *Inst. Min. and Met.*, vol. 15, 1905; p. 228; *Min. Jour.*, 1906, p. 128.
- STEPHAN, M. Einiges über die Erzeugung von Metallen im elektrischen Ofen. *Metall. und Erz*, vol. 1, 1912, p. 11.
- TAUSSIG, R. Large electric furnace. *Trans. Faraday Soc.*, vol. 5, 1909, p. 254.
- Present status of the development of large electric furnaces. *Met. and Chem. Eng.*, vol. 10, 1912, p. 686.
- VENATOR, W. Ueber Eisen Legierungen und Metall für die Stahl Industrie. *Stahl und Eisen*, vol. 28, 1908, p. 41; *Iron and Coal Trades Rev.*, vol. 76, 1908, p. 520.
- WEINTRAUB, E. Preparation and properties of pure boron. *Trans. Am. Electrochem. Soc.*, vol. 16, 1909, p. 165.

FURNACES.

- ADLER, E., and SABERSKY, E. A new electrical hardening furnace. *Trans. Faraday Soc.*, vol. 5, 1909, p. 15.
- FITZGERALD, F. A. J. Experiments with an electrothermic muffle furnace. *Electrochem. and Met. Ind.*, vol. 3, 1905, p. 135.
- The Borchers furnace. *Electrochem. and Met. Ind.*, vol., 3, 1905, p. 215.
- The Ruthenburg and Acheson furnaces. *Electrochem. and Met. Ind.*, vol. 3, 1905, p. 416.
- HARDEN, J. Some electric-furnace notes. *Met. and Chem. Eng.*, vol. 9, 1911, p. 130.
- JOHNSON, W. McA. Rotary electric furnace. *Electrochem. and Met. Ind.*, vol. 4, 1906, p. 321.
- SABERSKY, E. See Adler, E.
- TAYLOR, E. R. The manufacture of bisulphide of carbon in the electric furnace. *Trans. Am. Electrochem. Soc.*, vol. 1, 1902, p. 115.
- Some advances in the closed and continuous working electric furnace. *Trans. Am. Electrochem. Soc.*, vol. 2, 1902, p. 185.
- A closed electric furnace for reducing and distilling metals from their ores. *Trans. Am. Electrochem. Soc.*, vol. 11, 1907, p. 294.

GENERAL DESCRIPTION.

- BENNIE, P. McN. The electric furnace, its place in siderology. *Trans. Can. Min. Inst.*, vol. 13, 1910, p. 135.
- BROWN, O. W. The reduction of metal sulphides. *Trans. Am. Electrochem. Soc.*, vol. 9, 1906, p. 109.
- BURGESS, C. F. The present status of electric-furnace working. *Jour. West. Soc. Eng.*, vol. 10, 1905, p. 16.
- CATANI, R. The application of electricity in the metallurgy of Italy. *Jour. Iron and Steel Inst.*, vol. 84, No. 2, 1911, p. 215; *Met. and Chem. Eng.*, vol. 9, 1911, p. 642.
- DUSHMAN, S. Electrochemical and electrometallurgical developments in Canada. *Trans. Am. Electrochem. Soc.*, vol. 20, 1911, p. 419.
- FARNSWORTH, L. D. Experimental electric smelting. *Electrochem. and Met. Ind.*, vol. 6, 1908, p. 326.
- HARDEN, J. Some electric-furnace notes. *Met. and Chem. Eng.*, vol. 9, 1911, p. 130.
- HERING, C. Advantage of small high-speed electric furnaces. *Met. and Chem. Eng.*, vol. 11, 1913, p. 183.
- Elementary principles of the design and operation of electric furnaces. *Met. and Chem. Eng.*, vol. 8, 1910, p. 471.
- HUTTON, R. S. On the fusion of quartz in the electric furnace. *Trans. Am. Electrochem. Soc.*, vol. 2, 1909, p. 105.
- JOHNSON, W. McA., and SIEGER, G. N. Electric furnaces, their design, characteristics, and commercial application. *Met. and Chem. Eng.*, vol. 11, 1913, pp. 504, 563, 643.
- KEENEY, R. M., and LYON, D. A. Possible applications of the electric furnace to Western metallurgy. *Trans. Am. Electrochem. Soc.*, vol. 24, 1913, p. 118. *Met. and Chem. Eng.*, vol. 11, 1913, p. 577.
- KOWALKE, O. L. Electric-furnace conversion of iron pyrites into a magnetic form. *Trans. Am. Electrochem. Soc.*, vol. 13, 1908, p. 133.
- KUNYE, W., Beitrag zum Entwicklungstand neuerer elektroöfen. *Stahl und Eisen*, vol. 32, 1912, p. 1181.
- LOUVRIER, F. Electric heat vs. heat from fuel. *Electrochem. and Met. Ind.*, vol. 5, 1907, p. 298.
- Electric and fuel furnaces compared. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 159.

LYON, D. A. See Keeney, R. M.

METALLURGICAL AND CHEMICAL ENGINEERING. Editorial on the electric furnace and the scrap-metal business. Vol. 9, 1911, p. 620.

MINET, A. The electric furnace; its origin, transformation, and applications. Trans. Faraday Soc., vol. 1, 1905, p. 77; vol. 2, 1906, p. 1.

NERNST, W. Determination of vapor densities in an electric furnace. Trans. Am. Electrochem. Soc., vol. 3, 1903, p. 75.

RICHARDS, J. W. Conditions of progress in electrochemistry. Trans. Am. Electrochem. Soc., vol. 3, 1903, p. 59.

——— Electrochemistry at Sault Ste. Marie. Electrochem. Ind., vol. 1, 1902, p. 85.

——— Furnace efficiency. Electrochem. Ind., vol. 1, 1902, p. 46.

——— Niagara as an electrochemical center. Electrochem. Ind., vol. 1, 1902, p. 11.

——— The efficiency of electric furnaces. Trans. Am. Electrochem. Soc., vol. 2, 1902, p. 51.

——— The electrochemical industries of Norway. Trans. Am. Electrochem. Soc., vol. 20, 1911, p. 403.

——— The electric furnace in nonferrous metallurgy. Met. and Chem. Eng., vol. 8, 1910, p. 233.

——— What electrochemistry is accomplishing. Trans. Am. Electrochem. Soc., vol. 17, 1910, p. 75.

RICHARDS, J. W., and HERING, C. Efficiency—A discussion. Met. and Chem. Eng., vol. 9, 1911, p. 125.

RUTHENBURG, M. Electric smelting furnaces and their applications. Trans. Am. Electrochem. Soc., vol. 18, 1910, p. 185.

SIEGER, G. N. See Johnson, W. McA.

SNYDER, F. T. The reliability of electric furnaces for commercial work. Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 185.

WELLMAN, S. F. Discussion of the electric furnace. Iron Trade Rev., vol. 50, 1912, p. 1224.

GOLD AND SILVER.

CONKLIN, H. R. Electric furnace at Lluvia de Oro. Eng. and Min. Jour., vol. 93, 1912, p. 1189.

HEAT LOSSES.

CLEMENT, J. K., and EGY, W. L. Measurement of thermal conductivity of fire clay Bull. 36, Eng. Exper. Sta., University of Illinois.

EGY, W. L. See also Clement, J. K.

ELECTROCHEMICAL AND METALLURGICAL INDUSTRY. Heating furnace walls to reduce heat losses. Vol. 7, 1909, p. 494.

FITZGERALD, F. A. J. Experiments on heat insulation. Trans. Am. Electrochem. Soc., vol. 21, 1912, p. 535.

——— Heat losses in furnaces. Trans. Am. Electrochem. Soc., vol. 22, 1912, p. 111.

——— Heat losses of the electric furnace. Met. and Chem. Eng., vol. 9, 1911, p. 1270.

——— On the heat conductivity of carbon. Trans. Am. Electrochem. Soc., vol. 12, 1907, p. 165.

——— The use of carbon for the study of temperatures in the electric furnace. Trans. Am. Electrochem. Soc., vol. 6, 1904, p. 32.

HERING, C. Effects of the variations of thermal resistivities with the temperature. Trans. Am. Electrochem. Soc., vol. 21, 1912, p. 511.

——— Heat conductance and resistance of composite bodies. Electrochem. and Met. Ind., vol. 7, 1909, p. 11.

——— Heat conductance through walls of furnaces. Trans. Am. Electrochem. Soc., vol. 14, 1908, p. 215.

- HERRING, C. Improving the output and efficiency of existing electric furnaces. *Met. and Chem. Eng.*, vol. 8, 1910, p. 276.
- Method for determining thermal conductivities. *Trans. Am. Electrochem. Soc.*, vol. 18, 1910, p. 213.
- The thermal insulation of furnace walls. *Met. and Chem. Eng.*, vol. 10, 1912, p. 97.
- KREISINGER, HENRY. See Ray, W. T.
- METALLURGICAL AND CHEMICAL ENGINEERING. Electric-furnace temperature regulation. Vol. 8, 1910, p. 96.
- RAY, W. T., and KREISINGER, HENRY. The flow of heat through furnace walls. *Bull. 8, Bureau of Mines*, 1911, 32 pp.
- RANDOLPH, C. P. The thermal resistivity of insulating materials. *Trans. Am. Electrochem. Soc.*, vol. 21, 1912, p. 545.
- SAUNDERS, L. E. Temperature measurements on the silicon carbide furnace. *Trans. Am. Electrochem. Soc.*, vol. 21, 1912, p. 425.
- SNYDER, F. T. The flow of heat through furnace walls. *Trans. Am. Electrochem. Soc.*, vol. 18, 1910, p. 235.

INDUCTION FURNACE.

- ENGLEHARDT, V. Electric induction furnace for making steel. *Electrochem. and Met. Ind.*, vol. 3, 1905, p. 294.
- The induction furnace and its use in the steel industry. *Electrochem. and Met. Ind.*, vol. 6, 1908, p. 143.
- FITZGERALD, F. A. J. Experiments on melting in the induction furnace. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 10.
- GIN, G. Mathematics of the induction furnace. *Trans. Am. Electrochem. Soc.*, vol. 12, 1907, p. 97.
- HARDEN, J. Induction-furnace notes. *Met. and Chem. Eng.*, vol. 11, 1913, p. 559.
- Present status of the induction furnace. *Met. and Chem. Eng.*, vol. 11, 1913, p. 99.
- The efficiency of induction furnaces. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 320.
- HIARTH, A. Design of a 30-ton induction electric furnace. *Trans. Am. Electrochem. Soc.*, vol. 20, 1911, p. 293.
- NEUMANN, B., RÖCHLING, W., and RODENHAUSER, W. Induction furnace for three-phase currents. *Electrochem. and Met. Ind.*, vol. 6, 1908, p. 458.

LABORATORY FURNACES.

- BARD, E. F., and CALHANE, D. F. An efficient electric furnace for high temperatures. *Met. and Chem. Eng.*, vol. 10, 1912, p. 416.
- CALHANE, D. F., see Bard, E. F.
- ELECTROCHEMICAL AND METALLURGICAL INDUSTRY. Electric laboratory furnaces. Vol. 7, 1908, pp. 219, 222.
- Laboratory furnaces. Vol. 6, 1909, p. 256.
- Rheostat for small electric furnaces. Vol. 7, 1909, p. 168.
- FITZGERALD, F. A. J. Electric laboratory furnace. *Electrochem. and Met. Ind.*, vol. 3, 1906, p. 55.
- Experiments with an electrothermic muffle furnace. *Electrochem. and Met. Ind.*, vol. 3, 1905, p. 135.
- HUTTON, R. S., and PATTERSON, W. H. Electrically heated carbon tube furnaces. *Trans. Faraday Soc.*, vol. 1, 1905, p. 187.
- PATTERSON, W. H. See Hutton, R. S.
- WHITE, G. R. Laboratory resistance furnace. *Trans. Am. Electrochem. Soc.*, vol. 9, 1906, p. 143.

PIG IRON.

- BENNIE, P. McN. Electric-furnace pig iron in California. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 35.
- Application of the electric furnace to the metallurgy of iron and steel. *Electrochem. Ind.*, vol. 2, 1904, p. 307.
- CAMPBELL, D. E. Progress in the electrometallurgy of iron and steel. *Trans. Faraday Soc.*, vol. 7, 1912, p. 195.
- CARCANO, F. E. The production of pig iron in the electric furnace and the industrial utilizations of pyrite residue. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 155.
- CATANI, R. Large electric furnaces in the electrometallurgy of iron and steel. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909; p. 159; *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 268.
- Reduction of iron ore in the electric furnace. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 153.
- CRAWFORD, J. Progress of electric smelting at Heroult, California. *Met. and Chem. Eng.*, vol. 11, 1913, p. 386.
- ELECTROCHEMICAL AND METALLURGICAL INDUSTRY. Iron reduction at the "Soo" by the Héroult electric furnace process. Vol. 4, 1906, p. 124.
- Electric iron and steel industry in Canada and in Sweden and Norway. Vol. 7, 1909, p. 419.
- FRICK, O. Electric reduction of iron ores with special reference to results obtained in electro-metals furnace at Trollhättan, Sweden, and Noble furnace at Heroult, California. *Met. and Chem. Eng.*, vol. 9, 1911, p. 631.
- GIN, G. On the electrical reduction of titaniferous iron ores. *Trans. Am. Electrochem. Soc.*, vol. 11, 1907, p. 291.
- GREENE, A. E., and MACGREGAR, F. S. On the electrothermic reduction of iron ores. *Trans. Am. Electrochem. Soc.*, vol. 12, 1907, p. 65.
- HAANEL, E. Investigation of an electric shaft furnace, Domnarfvet, Sweden. *Trans. Faraday Soc.*, vol. 5, 1909, p. 306.
- Preliminary report on the experiments made at Sault Ste. Marie under Government auspices on the smelting of Canadian iron ores by the electrothermic process. *Trans. Faraday Soc.*, vol. 2, 1906, p. 120.
- The electric shaft furnace of the Aktiebolaget Elektrometall, Ludvika, Sweden. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 25.
- HARBORD, F. W. Recent developments in electric smelting in connection with iron and steel. *Trans. Faraday Soc.*, vol. 1, 1905, p. 140.
- HARDEN, J. Smelting iron ores in the electric furnace in comparison with blast-furnace practice. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 16.
- KEENEY, R. M., and LYON, D. A. Possible applications of the electric furnace to western metallurgy. *Trans. Am. Electrochem. Soc.*, vol. 24, 1913, p. 118; *Met. and Chem. Eng.*, vol. 11, 1913, p. 577.
- KELLER, C. A. Contribution to the study of electric furnaces as applied to the manufacture of iron and steel. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 87.
- Electrothermics of iron and steel. *Trans. Faraday Soc.*, vol. 2, 1906, p. 36.
- LEFFLER, J. A., and NYSTRÖM, E. Electric furnace pig iron at Trollhättan. *Met. and Chem. Eng.*, vol. 10, 1912, p. 413.
- LEFFLER, J. A., and ODELBERG, E. The electric iron reduction plant at Trollhättan, Sweden. *Met. and Chem. Eng.*, vol. 9, 1911, pp. 368, 459, and 505.
- LYON, D. A. The electric furnace in the production of pig iron from ore. *Met. and Chem. Eng.*, vol. 11, 1913, p. 15.
- The Noble Electric Steel Company's plant. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 39.
- The use of electric-furnace pig iron in the open-hearth furnace. *Met. and Chem. Eng.*, vol. 10, 1912, p. 539.
- See also Keeney, R. M.

- LYMAN, J. The electric furnace for the manufacture of iron and steel. *Trans. Am. Electrochem. Soc.*, vol. 19, 1910, p. 193.
- MACGREGAR, F. S. See Greene, A. E.
- NEUMANN, B. Electric-furnace methods in iron and steel manufacture in comparison with ordinary metallurgical processes. *Electrochem. Ind.*, vol. 2, 1904, p. 488.
- NYSTRÖM, E. See Leffler, J. A.
- OELBERG, E. See Leffler, J. A.
- RICHARDS, J. W. Discussion of the experiments made at Sault Ste. Marie on the electrical reduction of iron ores. *Trans. Am. Electrochem. Soc.*, vol. 12, 1907, p. 81.
- Electric reduction of iron ore. *Jour. Franklin Inst.*, vol. 169, 1910, p. 131.
- Gas circulation in electric reduction furnaces. *Trans. Am. Electrochem. Soc.*, vol. 21, 1912, p. 403.
- Metallurgical calculations; the electrometallurgy of iron and steel. *Electrochem. and Met. Ind.*, vol. 5, 1907, p. 165.
- The electrometallurgical revolution in iron and steel industry of Norway and Sweden. *Proc. Eng. Soc. West. Pa.*, vol. 27, 1911, p. 125.
- The electrothermic production of iron and steel, method and furnace. *Jour. Franklin Inst.*, vol. 164, 1907, p. 443; vol. 165, 1908, p. 47.
- The electric-furnace reduction of iron ore. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 53.
- ROBERTSON, T. D. Recent progress in electrical iron smelting in Sweden. *Trans. Am. Electrochem. Soc.*, vol. 20, p. 375.
- ROSSI, A. J. Electric-smelting and blast-furnace gases. *Electrochem. and Met. Ind.*, vol. 3, 1904, pp. 150, 190.
- Note on the utilization of blast-furnace gases in connection with the electric smelting of iron. *Trans. Am. Electrochem. Soc.*, vol. 7, 1905, p. 199.
- SIMPSON, L. The electric reduction of iron ores and the conversion of iron into steel in an electric furnace. *Electrochem. Ind.*, vol. 1, 1903, p. 277.
- STANSFIELD, A. Laboratory experiments on removing silicon and phosphorus in smelting iron ores. *Trans. Can. Min. Inst.*, vol. 10, 1907, p. 128.
- TAYLOR, E. R. Contribution to the electric smelting of iron ore. *Trans. Am. Electrochem. Soc.*, vol. 16, 1909, p. 229.
- YNGSTROM, L. Electric blast furnace at Domnarfvet, Sweden. *Met. and Chem. Eng.*, vol. 8, 1910, p. 11.
- Electric production of iron from iron ore at Domnarfvet. *Engineering (London)*, vol. 109, 1910, pp. 206, 234.

POWER.

- ASHCROFT, E. A. The influence of cheap electricity on electrolytic and electrothermal industries. *Trans. Faraday Soc.*, vol. 4, 1908, pp. 134, 149.
- CRABTREE, F. Possibilities of cheap power in the Pittsburgh district. *Trans. Am. Electrochem. Soc.*, vol. 17, 1910, p. 95.
- ELECTROCHEMICAL AND METALLURGICAL INDUSTRY. Regulation of current in electric furnaces. Vol. 3, 1905, p. 9.
- JONES, A. J. Transforming stations of Niagara electrochemical and electrometallurgical industries. *Trans. Am. Electrochem. Soc.*, vol. 20, 1911, p. 455.
- KNESCHE, F. A. Electric smelting of ore in the United States (an investigation of the possibility of utilizing energy from waste fuels). *Iron Trade Rev.*, vol. 48, 1911, p. 65.
- LUCKE, C. E. Power cost. *Trans. Am. Electrochem. Soc.*, vol. 11, 1907, p. 339.
- MEYER, J. Electrochemical processes as station-load equalizers. *Trans. Am. Electrochem. Soc.*, vol. 11, 1907, p. 369.
- Power for electrochemical purposes. *Trans. Am. Electrochem. Soc.*, vol. 14, 1908, p. 377.

- METALLURGICAL AND CHEMICAL ENGINEERING. The power problem for electrochemical plants. Vol. 9, 1911, p. 384.
- NORTHROP, E. F. A new type of ammeter for the accurate measurement of alternating currents above 1,000 amperes. Trans. Am. Electrochem. Soc., vol. 15, 1909, p. 303.
- RANDALL, K. C. A study in heavy alternating-current conductors for electric furnaces. Trans. Am. Electrochem. Soc., vol. 17, 1910, p. 139.
- SCOTT-HANSEN, A. Hydroelectric plants in Norway and their application to electrochemical industries. Trans. Faraday Soc., vol. 7, 1911, p. 78.
- SIMPSON, L. Cost of electric power used in electric reduction works. Electrochem. Ind., vol. 2, p. 421.
- SPERRY, E. A. Electrochemical processes as station-load equalizers. Trans. Am. Electrochem. Soc., vol. 9, 1906, p. 147.
- Utilization of power stations for electrochemical and electrothermal processes during periods of low load. Trans. Am. Electrochem. Soc., vol. 14, 1908, p. 259.
- SYKES, W. Power supply to electric furnaces for refining iron and steel. Trans. Am. Electrochem. Soc., vol. 21, 1912, p. 383.
- WILSON, J. R. High-tension equipment for electrochemical plants. Trans. Am. Electrochem. Soc., vol. 21, 1912, p. 371.

REFRACTORY MATERIALS.

- BLEININGER, A. V., and BROWN, G. H. Testing of clay refractories, with special reference to their load-carrying capacities. Bureau of Standards Technologic Paper 7, 1911, 78 pp.
- BROWN, G. H. See Bleininger, A. V.
- JOURNAL OF INDUSTRIAL AND ENGINEERING CHEMISTRY. Committee on specifications for refractories of Institute of Gas Engineers. Vol. 4, 1912, p. 618.
- FITZGERALD, F. A. J. Refractory materials in electrical resistance furnaces. Electrochem. Ind., vol. 2, 1904, p. 439.
- HARBISON-WALKER REFRACTORIES CO. Refractories in furnace construction. Met. and Chem. Eng., vol. 8, 1910, p. 106.
- KANOLT, C. W. Melting points of fire bricks. U. S. Bureau of Standards Technologic Paper 10, 1912, 17 pp.
- MCLEOD, D. L., STANSFIELD, A., and MCMAHON, J. W. The electrical resistivity of fire bricks at high temperatures. Trans. Am. Electrochem. Soc., vol. 22, 1912, p. 89.
- MCMAHON, J. W. See McLeod, D. L.
- QUENEAU, A. L., and WOLGODINE, S. Conductivity, porosity, and gas permeability of refractory materials. Electrochem. and Met. Ind., vol. 7, 1909, p. 383.
- RIGG, G. Defects in refractory brick and their causes. Met. and Chem. Eng., vol. 8, 1910, p. 237.
- STANSFIELD, A. See McLeod, D. L.
- WOLGODINE, S. See Queneau, A. L.

RESISTANCE FURNACE.

- ALEXANDER, W. A., TUCKER, S. A., and HUDSON, H. K. Relative efficiency of the arc and resistance furnace for the manufacture of calcium carbide. Trans. Am. Electrochem. Soc., vol. 15, 1909, p. 411.
- CAUCHOIS, R. W., DOTY, A., and TUCKER, S. A. Granular-carbon resistors. Trans. Am. Electrochem. Soc., vol. 12, 1907, p. 171.
- COLLENS, C. L. Some principles of resistor-furnace design. Trans. Am. Electrochem. Soc., vol. 9, 1906, p. 31.
- DEMPSTER, J. T. K. Alloys for resistors. Electrochem. and Met. Ind., vol. 8, 1910 p. 14.

ELECTROCHEMICAL AND METALLURGICAL INDUSTRY. The works of the International Acheson Graphite Co. at Niagara Falls. Vol. 7, 1909, p. 187.

DOTY, A. See also Cauchois, R. W.

FITZGERALD, F. A. J. A new electric resistance furnace. Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 273; Met. and Chem. Eng., vol. 9, 1911, p. 259.

——— A new resistor furnace. Met. and Chem. Eng., vol. 8, 1910, p. 317.

——— Industrial resistance furnaces. Electrochem. and Met. Ind., vol. 3, 1905, p. 297.

——— Materials for resistors. Electrochem. Ind., vol. 2, 1904, p. 490.

——— Miscellaneous accessories of resistor furnaces. Electrochem. and Met. Ind., vol. 3, 1905, p. 9.

——— Note on an unsuccessful furnace experiment. Trans. Am. Electrochem. Soc., vol. 20, 1911, p. 281.

——— Note on some theoretical considerations in the construction of resistance furnaces. Trans. Am. Electrochem. Soc., vol. 4, 1903, p. 9.

——— Refractory materials in electrical resistance furnaces. Electrochem. Ind., vol. 2, 1904, p. 439.

——— Resistance furnace for crucibles. Electrochem. and Met. Ind., vol. 3, 1905, p. 55.

——— Some first principles of electrical resistance furnaces. Electrochem. Ind., vol. 2, 1904, p. 342.

——— The carborundum furnace. Electrochem. and Met. Ind., vol. 4, 1906, p. 53.

GOODWIN, J. H. Granular-carbon resistance furnaces. Met. and Chem. Eng., vol. 9, 1911, p. 188.

HERING, C. A new type of electric furnace. Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 255.

——— Electric furnaces for molten materials. Met. and Chem. Eng., vol. 9, 1911, p. 371.

HEUMANN, E. M., KUDLICH, H. F., and TUCKER, S. A. The preparation of silundum. Trans. Am. Electrochem. Soc., vol. 16, 1909, p. 207.

HUDSON, H. K. See Alexander, W. A.

KUDLICH, H. F. See Heumann, E. M.

TUCKER, S. A. See also Alexander, W. A.; Cauchois, R. W.; and Heumann, E. M.

WATTS, O. P. An electric furnace for heating crucibles. Electrochem. and Met. Ind., vol. 4, 1906, p. 273.

STEEL.

AMBERG, R. Deoxidation and desulphurization in electric steel furnaces. Electrochem. and Met. Ind., vol. 7, 1909, p. 115.

——— The function of slag in electric steel refining. Trans. Am. Electrochem. Soc., vol. 22, 1912, p. 133; Met. and Chem. Eng., vol. 10, 1912, p. 601.

ARNOU, M. G. Notes on the direct reduction of iron ores in the electric furnace. Rev. de Métal., vol. 7, 1910, p. 1190.

BAILY, T. F. An electric furnace for heating bars and billets. Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 285.

——— Further development of electric furnace for heating bars and billets. Trans. Am. Electrochem. Soc., vol. 21, 1912, p. 419.

BENNIE, P. McN. Application of the electric furnace to the metallurgy of iron and steel. Electrochem. Ind., vol. 2, 1904, p. 307.

BORCHERS, W. Electric smelting with the Girod furnace. Trans. Am. Inst. Min. Eng., vol. 41, 1910, p. 120.

BOWMAN, R. G., and DITTUS, E. J. The direct production of molybdenum steel in the electric furnace. Trans. Am. Electrochem. Soc., vol. 20, 1911, p. 355.

BURGESS, C. F. Electrolytic refining as a step in the production of steel. Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 181.

- CAMPBELL, D. F. Electric steel refining. *Jour. Iron and Steel Inst.*, vol. 82, No. 2, 1910, p. 197.
- Progress in the electrometallurgy of iron and steel. *Trans. Faraday Soc.*, vol. 7, 1912, p. 195.
- CATANI, R. Large electric furnaces in the electrometallurgy of iron and steel. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 159; *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 268.
- The application of electricity in the metallurgy of Italy. *Jour. Iron and Steel Inst.*, vol. 84, No. 2, 1911, p. 215; *Met. and Chem. Eng.*, vol. 9, 1911, p. 642.
- CLARK, C. B. Various types and applications of electric steel furnaces. *Met. and Chem. Eng.*, vol. 10, 1912, p. 373.
- DITTUS, E. J. See Bowman, R. G.
- Electrochemical and Metallurgical Industry. A Swiss electric-furnace steel plant. Vol. 6, 1908, p. 452.
- Electric iron and steel industry in Canada and in Sweden and Norway. Vol. 7, 1909, p. 419.
- Removal of sulphur in electric steel furnaces. Vol. 6, 1908, p. 405.
- The electric furnace in iron and steel metallurgy. Vol. 5, 1907, p. 24.
- The Girod electric steel furnace. Vol. 6, 1908, p. 428.
- Electrochemical Industry. Steel production in the electric furnace. Vol. 1, 1902, p. 247.
- ENGLEHARDT, V. Electric induction furnace for making steel. *Electrochem. and Met. Ind.*, vol. 3, 1905, p. 294.
- The induction furnace and its use in the steel industry. *Electrochem. and Met. Ind.*, vol. 6, 1908, p. 143.
- EVANS, J. W. Laboratory experiments on the electric smelting of iron ore. *T. Can. Min. Inst.*, vol. 9, 1906, p. 128.
- FITZGERALD, F. A. J. The application of the Lash process to the electric furnace. *Ann. Am. Electrochem. Soc.*, vol. 15, 1909, p. 149.
- The Lash steel process and the electric furnace. *Ann. Am. Electrochem. Soc.*, vol. 14, 1908, p. 239.
- GIN, G. Automatically circulating furnaces of the Gin type for the electrical production of steel. *Trans. Faraday Soc.*, vol. 5, 1909, p. 137.
- Calculations of a Gin self-circulating induction steel furnace. *Ann. Am. Electrochem. Soc.*, vol. 15, 1909, p. 215.
- Note on recent developments in the Gin electric steel furnace. *Trans. Faraday Soc.*, vol. 2, 1906, p. 44.
- The new Gin process for the electrical manufacture of steel. *Trans. Am. Electrochem. Soc.*, vol. 8, 1905, p. 105.
- The self-circulating Gin furnace for the electric manufacture of steel. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 205.
- GIROD, P. Studies in the electrometallurgy of ferro-alloys and steels. *Trans. Faraday Soc.*, vol. 6, 1910, pp. 172-184.
- The electric steel furnace in foundry practice. *Met. and Chem. Eng.*, vol. 10, 1912, p. 663.
- The Girod electric furnace for the manufacture of steel. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 127; *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 259.
- GREENE, A. E. Electric heating and the removal of phosphorus from iron. *Trans. Am. Electrochem. Soc.*, vol. 22, 1912, p. 123.
- Electric steel processes as competitors of the Bessemer and open hearth. *Trans. Am. Electrochem. Soc.*, vol. 19, 1911, p. 233.
- HARBORD, F. W. Recent developments in electric smelting in connection with iron and steel. *Trans. Faraday Soc.*, vol. 1, 1905, p. 140.

- HARDEN, J. A new electric furnace for steel melting and refining. *Met. and Chem. Eng.*, vol. 9, 1911, p. 38.
- Present status of the induction furnace, *Met. and Chem. Eng.*, vol. 9, 1913, p. 99.
- Recent developments of the Kjellin and Röchling-Rodenhauser electric induction furnaces. *Trans. Faraday Soc.*, vol. 4, 1908, p. 120.
- The "Paragon" electric furnace and recent developments in metallurgy. *Met. and Chem. Eng.*, vol. 9, 1911, p. 595; *Trans. Faraday Soc.*, vol. 7, 1912, p. 183.
- HERING, C. Possible reduction of the power consumption in electric steel-refining furnaces. *Met. and Chem. Eng.*, vol. 9, 1911, p. 590.
- HÉROULT, P. L. T. Electric steel process. *Electrochem. Ind.*, vol. 1, 1903, p. 449.
- Electric steel refining. *Met. and Chem. Eng.*, vol. 10, 1912, p. 601.
- The electrometallurgy of iron and steel. *Trans. Am. Electrochem. Soc.*, vol. 6, 1904, p. 129.
- The Héroult furnace. *Electrochem. and Met. Ind.*, vol. 5, 1907, p. 411.
- HIBBARD, H. D. The present value of electrical structural steel. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 231.
- IZART, J. The Stassano furnace installed at the new works in Turin. *Electrician*, Oct. 12, 1907.
- KEARNS, J. E. Electric annealing furnaces. *Electrochem. and Met. Ind.*, vol. 4, 1906, p. 95.
- KEENEY, R. M. The production of steels and ferro-alloys directly from ore in the electric furnace. *Jour. Iron and Steel Inst.*, Carnegie Scholarship Memoirs, vol. 4, 1912, p. 108.
- KEENEY, R. M., and LEE, G. M. The direct production of steels and ferro-alloys from ore in the electric furnace. *West. Chem. and Metal.*, vol. 6, 1910, pp. 269, 323, 347.
- KELLER, C. A. A contribution to the study of electric furnaces as applied to the manufacture of iron and steel. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 87; *Trans. Faraday Soc.*, vol. 5, 1909, p. 113.
- Electrothermics of iron and steel. *Trans. Faraday Soc.*, vol. 2, 1906, p. 36.
- KERSHAW, J. B. C. Electric-furnace manufacture of steel. *Iron Trade Rev.*, vol. 51, 1912, pp. 865, 959, 1007, 1067, 1105, 1169.
- KJELLIN, F. A. The Kjellin and Röchling-Rodenhauser electric furnace. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 173.
- LEE, G. M. See Keeney, R. M.
- LIPIN, W. The Nathusius electric steel furnace. *Met. and Chem. Eng.*, vol. 10, 1912, p. 227.
- LYMAN, J. The electric furnace for the manufacture of iron and steel. *Trans. Am. Electrochem. Soc.*, vol. 19, 1910, p. 193.
- METALLURGICAL AND CHEMICAL ENGINEERING. South Chicago electric-furnace plant of the United States Steel Corporation. Vol. 8, 1910, p. 179.
- The electric furnace for small steel castings. Vol. 10, 1912, p. 54.
- MOLDENKE, R. Electric melting for the foundry. *Electrochem. and Met. Ind.*, vol. 5, 1907, p. 42.
- MUELLER, A. The manufacture of steel in the Girod electric furnace. *Met. and Chem. Eng.*, vol. 9, 1911, p. 581.
- NATHUSIUS, H. Improvements in electric furnaces and their application in the manufacture of steel. *Jour. Iron and Steel Inst.*, vol. 135, No. 1, 1912, p. 57.
- NEUMANN, B. Electric-furnace methods in iron and steel manufacture in comparison with the ordinary metallurgical processes. *Electrochem. Ind.*, vol. 2, 1914, p. 488.
- Röchling-Rodenhauser induction furnace for three-phase currents. *Electrochem. and Met. Ind.*, vol. 6, 1908, p. 458.
- OSBORNE, C. G. A few experiments with the 15-ton Héroult electric furnace at South Chicago. *Trans. Am. Electrochem. Soc.*, vol. 19, 1911, p. 205.

- PERKINS, F. C. Kjellin electric furnace at Gysinge, Sweden, for the manufacture of steel. *Electrochem. Ind.*, vol. 1, 1903, p. 576.
- QUENEAU, A. L. A new electric steel furnace. *Trans. Am. Electrochem. Soc.*, vol. 17, 1910, p. 131.
- RICHARDS, J. W. Metallurgical calculations; the electrometallurgy of iron and steel. *Electrochem. and Met. Ind.*, vol. 5, 1907, p. 165.
- Pig steel made directly from ore in the electric furnace. *Met. and Chem. Eng.*, vol. 10, 1912, p. 397.
- The electrothermic production of iron and steel, methods and furnaces. *Jour. Franklin Inst.*, vol. 164, 1907, p. 443; vol. 165, 1908, p. 47.
- The electrometallurgical revolution in the iron and steel industry of Norway and Sweden. *Proc. Eng. Soc. West. Pa.*, vol. 27, 1911, p. 125.
- The Hiorth electric steel furnace. *Trans. Am. Electrochem. Soc.*, vol. 18, 1910, p. 191.
- The largest electric steel works. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 9.
- The passing of crucible steel. *Met. and Chem. Eng.*, vol. 8, 1910, p. 563.
- ROBERTSON, T. D. The Grönwall steel-refining furnace. *Met. and Chem. Eng.*, vol. 9, 1911, p. 573.
- ROBINSON, T. W. Electric steel-furnace experience at South Chicago. *Met. and Chem. Eng.*, vol. 10, 1912, p. 372.
- RODENHAUSER, W. The electric furnace and electric process of steel making. *Jour. Iron and Steel Inst.*, vol. 79, 1909, No. 1, p. 261.
- ROWLANDS, T. Induction-furnace progress. *Trans. Am. Electrochem. Soc.*, vol. 17, 1910, p. 103.
- STASSANO, E. Note on the rotating electric steel furnace in the artillery construction works, Turin. *Trans. Faraday Soc.*, vol. 2, 1906, p. 150.
- The application of the electric furnace to siderurgy. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 63.
- Treatment of iron and steel in the electric furnace. *Electrochem. and Met. Ind.*, vol. 6, 1908, p. 315.
- STOUGHTON, B. Notes on iron and steel and electrothermic manufacture of iron and steel. *Jour. Franklin Inst.*, vol. 167, 1909, p. 73.
- THALLNER, O. The manufacture of high-grade steel in the electric furnace. *Electrochem. and Met. Ind.*, vol. 6, 1908, p. 26.
- TURNBALL, R. The Héroult electric steel furnace. *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 139; *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 260.
- VOM BAUR, C. H. Electric induction and resistance furnaces for steel. *Trans. Am. Electrochem. Soc.*, vol. 22, 1912, p. 117.
- The Röchling-Rodenhauser furnace of the Crucible Steel Casting Co., Lansdowne, Pa. *Iron Trade Rev.*, vol. 52, 1913, p. 153; *Met. and Chem. Eng.*, vol. 11, 1913, p. 113.
- WALKER, W. R. Electric furnace as a possible means of producing an improved quality of steel. *Met. and Chem. Eng.*, vol. 10, 1912, p. 371.
- WEDDING, H. A modification of the induction furnace for steel refining. *Electrochem. and Met. Ind.*, vol. 6, 1908, p. 10.
- WELLMAN, S. I. Electric steel for rails. *Met. and Chem. Eng.*, vol. 10, 1912, p. 372.

TIN.

- HARDEN, J. Electric tin smelting. *Met. and Chem. Eng.*, vol. 9, 1911, p. 453.
- WILE, R. S. An electric furnace for the treatment of tin dross, concentrates from cyanide mills, and other metallurgical works. *Met. and Chem. Eng.*, vol. 10, 1912, p. 495.
- Reduction of tin dross in an electric furnace. *Trans. Am. Electrochem. Soc.*, vol. 18, 1910, p. 205.

VACUUM AND HIGH-PRESSURE FURNACE.

- ARSEM, W. C. Electric vacuum furnace installation in research laboratory of the General Electric Co. *Ind. and Eng. Chem.*, vol. 2, 1910, p. 3.
- The electric vacuum furnace. *Trans. Am. Electrochem. Soc.*, vol. 9, 1906, p. 153.
- FINK, C. G. Electric vacuum furnace installation. *Electrochem. and Met. Ind.*, vol. 4, 1906, p. 223; vol. 7, 1908, p. 428; vol. 8, 1909, p. 96.
- Vacuum-furnace metallurgy. *Trans. Am. Electrochem. Soc.*, vol. 21, 1912, p. 445.
- HUTTON, R., and PETAVEL, M. Electric-furnace reactions under high pressure. *Electrochem. and Met. Ind.*, vol. 6, 1908, p. 97.
- PETAVEL, M. See Hutton, R.

ZINC AND LEAD.

- BETTS, A. G. Electric lead smelting. *Electrochem. and Met. Ind.*, vol. 4, 1906, p. 169.
- BLEECKER, W. F. An electrolytic method for the reduction of blue powder. *Trans. Am. Electrochem. Soc.*, vol. 21, 1912, p. 359.
- BROWN, O. W. The reduction of metal sulphides. *Trans. Am. Electrochem. Soc.*, vol. 9, 1906, p. 109.
- BROWN, O. W., and OESTERLE, W. F. The electric smelting of zinc. *Trans. Am. Electrochem. Soc.*, vol. 8, 1905, p. 170.
- ENGINEERING AND MINING JOURNAL. Electric zinc smelting. Vol. 94, 1912, p. 1109.
- FITZGERALD, F. A. J. A new electric resistance furnace. *Trans. Am. Electrochem. Soc.*, vol. 19, 1911, p. 273; *Met. and Chem. Eng.*, vol. 9, 1911, p. 259.
- Note on an unsuccessful furnace experiment. *Trans. Am. Electrochem. Soc.*, vol. 20, 1911, p. 281.
- FLEURVILLE, E. Electric zinc smelting. *Electrochem. and Met. Ind.*, vol. 7, 1909, p. 468.
- GIN, G. The electrometallurgy of zinc. *Trans. Am. Electrochem. Soc.*, vol. 12, 1907, p. 117.
- HARBORD, F. W. Zinc smelting at Trollhättan. *Eng. and Min. Jour.*, vol. 93, 1912, p. 344.
- INGALLS, W. R. The electric smelting of zinc ore. *Eng. and Min. Jour.*, vol. 94, 1912, p. 7; *Met. and Chem. Eng.*, vol. 10, 1912, p. 481; *Trans. Can. Min. Inst.*, vol. 15, 1912, p. 101.
- JOHNSON, C. F. The electric zinc-smelting furnace. *Met. and Chem. Eng.*, vol. 10, 1912, p. 28.
- JOHNSON, W. McA. Notes on electric zinc smelting. *Met. and Chem. Eng.*, vol. 10, 1912, p. 537.
- The art of electric zinc smelting. *Trans. Am. Electrochem. Soc.*, vol. 24, 1913, p. 191; *Met. and Chem. Eng.*, vol. 11, 1913, p. 582.
- The electrometallurgy of zinc and its relation to present practice. *Trans. Am. Electrochem. Soc.*, vol. 17, 1910, p. 265.
- KEENEY, R. M., and LYON, D. A. Possible applications of the electric furnace to western metallurgy. *Trans. Am. Electrochem. Soc.*, vol. 24, 1913, p. 119; *Met. and Chem. Eng.*, vol. 11, 1913, p. 577.
- KOWALKE, O. L. The volatility of zinc oxide. *Trans. Am. Electrochem. Soc.*, vol. 21, 1912, p. 557.
- LOUVRIER, F. Electric zinc smelting. *Met. and Chem. Eng.*, vol. 11, 1913, p. 603.
- Causes of the practical nonsuccess of the electric furnace in treating zinc ores. *Met. and Chem. Eng.*, vol. 10, 1912, p. 747.
- LYON, D. A. See Keeney, R. M.

- METALLURGICAL AND CHEMICAL ENGINEERING. Editorial on the condensation problem in electric-zinc smelting, vol. 10, 1912, p. 451.
- Electric-zinc smelting in Norway and Sweden. Vol. 9, 1911, p. 673.
- OESTERLE, W. F. See Brown, O. W.
- PETERSON, P. E. The electric-zinc furnace. Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 215; Met. and Chem. Eng., vol. 11, 1913, p. 583.
- RICHARDS, J. W. The electric furnace in nonferrous metallurgy. Met. and Chem. Eng., vol. 8, 1910, p. 233.
- The Johnson electric-zinc furnace. Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 311.
- SNYDER, T. F. The condensation of zinc vapor from electric furnaces. Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 317; Met. and Chem. Eng., vol. 9, 1911, p. 265.
- VOGEL, J. L. F. The electrolysis of fused zinc chloride in cells heated externally. Trans. Faraday Soc., vol. 2, 1906, p. 56.
- WEEKS, C. A. Melting nonferrous metals in an electric furnace. Met. and Chem. Eng., vol. 9, 1911, p. 363.

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